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GATE MN

2025

FORMULA BOOK

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GATE Mining Formula Notes

By the help of mining gyan

VINOD CHANDRAKAR SIR

and faculty

~Mrityunjay Awasthi

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* Mining Method *

NOTES —: Formula :— DATE

01). **Percentage of extraction :-**

$$R = \frac{(a_1 + b_1)(a_2 + b_2) - a_1 \times a_2}{(a_1 + b_1) \times (a_2 + b_2)}$$

For Square pillar (extraction Ratio)

$$R = 1 - \frac{a^2}{(a+b)^2}$$

a_1, a_2 = length and width of pillars

b_1, b_2 = Size of gallaries in dip and Rise direction.

02). **Tributary area :-**

$$TA = (a_1 + b_1) \times (a_2 + b_2) - a_1 \times a_2$$

03). **Load acting on pillar :-**

$$= Y \times D \times \frac{1}{1-R} \Rightarrow Y \times D \times \frac{(a+b)^2}{a^2}$$

Y = weight per unit volume.
D = Depth,
R = Extraction Ratio.

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04). **Strength of pillar :-**

$$S = 1320 \times \frac{W^{0.46}}{H^{0.66}} \text{ lb/m}^2$$

1 lb = 454 Kg.
1 inch = 0.0254 meter.

W = width of pillar.
H = height of pillar.

$$S = 7.2 \times \frac{W^{0.46}}{H^{0.66}} \text{ MPa.}$$

05). **Factor of Safety (FOS) :-**

$$FOS = \frac{\text{Strength of Pillar}}{\text{Load acting on pillar } \left[\gamma \times D \times \frac{(a+b)^2}{a^2}\right]}$$

06). **Relative weight Strength (RWS_{exp}) :-**

$$RWS_{exp} = \frac{AWS_{exp}}{AWS_{ANFO}} \times 100$$

AWS = Absolute weight strength (MJ/Kg)

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07). **Absolute bulk Strength :-**

$$ABS_{exp} = ANS_{exp} \times P_{exp}$$

08). **Relative bulk Strength (RBS) :-**

$$RBS_{exp} = \frac{ABS_{exp}}{ABS_{ANFO}} \times 100$$

09). **Powder Factor :-**

$$= \frac{\text{Coal produced (in tonne)}}{\text{explosive used (in kg)}}$$

10). **Daubriche experiment for VOD :-**

$$VOD_{exp} = VOD_{known} \times \frac{d_{exp}}{2d\,(\text{on plate})}$$

11). **Detonation Pressure :-**

$$P = \frac{1}{4} \rho \times (VOD)^2 \text{ MPa}$$

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12). **Output per man Shift (OMS) :-**

$$OMS = \frac{\text{Output per shift}}{\text{total man power per shift}}$$

13). **Number of faces :-**

Maximum = $3n-2$
Average = $2n-1$
minimum = n

n = number of headings

14). **Power** = $\frac{\text{Energy (work)}}{\text{time}}$

15). **Thickness of Immediate Roof (H) :-**

$$H = \frac{t}{K-1}$$

t = mining height, K = Bulking factor

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16). **Tonnage factor :-**

$$TF = \frac{Volume}{weight}$$

$$Tonnage = \frac{weight}{Volume}$$

17). **Stripping Ratio :-**

$$SR = \frac{Volume\ of\ OB\ (m^3)}{weight\ of\ mineral\ (tn)}$$

18). **Area of parallelogram :-**

area of parallelogram = base × height

19). **Break even Stripping Ratio :-**

$$BESR = \frac{\text{Recoverable price per tonne of ore} - \left(\text{operating cost per tonne of ore} + \text{profit margin per tonne of ore}\right)}{\text{Cost of extraction per } m^3 \text{ of OB / (stripping cost)}}$$

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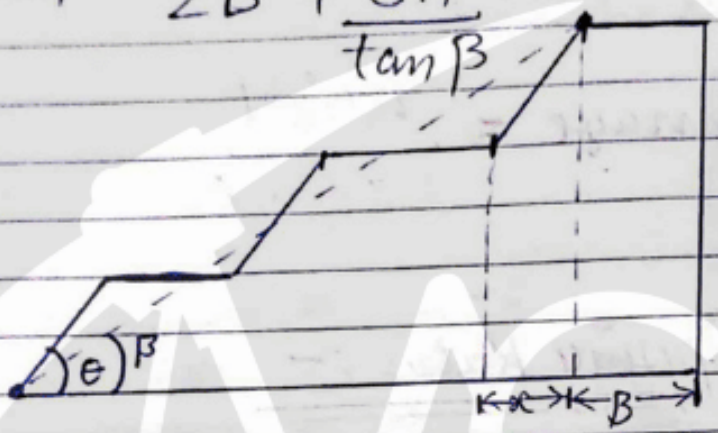
20). Overall slope angle :-

$$\tan\beta = \frac{3H}{2B + \frac{3H}{\tan\beta}}$$



$$x = \frac{H}{\tan\theta}$$

$$\beta = \tan^{-1}\left(\frac{3H}{2B + \frac{3H}{\tan\theta}}\right)$$

21). Match factor :-

$$= \frac{\text{Arrival rate}}{\text{Service rate}} \Rightarrow \frac{\text{No. of trucks} \times \text{loaded cycle time}}{\text{truck cycle time} \times \text{No. of loaders}}$$

22). Spherical charge :-

$\frac{L}{D} \leq 6$, L = Length of explosive column.
D = Dia of hole.

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23). **Peak particle Velocity (PPV) :-**

$$K = \left(\frac{D}{\sqrt{Q}}\right)^E$$

K = Site constant

D = Distance of transducer (starting point to measuring point)

Q = maximum charge weight per delay.

24). **Dilution :-**

$$= \frac{V_{waste}}{V_{insitu\ ore}}$$

25). **Max. roof tension :-**

$$\sigma_t = \frac{3}{4} P \left(\frac{L}{h}\right)^2 = \frac{3}{4} \times (\gamma \times h) \times \left(\frac{L}{h}\right)^2$$

γ = Specific weight.

h = Strata thickness.

L = Entry length.

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26). **Roof Sag :-**

$$= \left(\frac{5}{32}\right)\left(\frac{PL^4}{Eh^3}\right)$$

27). **Protodyakonov index :-**

$$= \frac{\sigma_c}{100}$$

σ_c = uniaxial compressive strength.

28). **Critical width :-**

$$W = 2H\tan\alpha$$

29). **Support load :-**

$$= \gamma \times B \times \left(\frac{100 - RMR}{100}\right)$$

B = Width of gallery.

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30). **Impedence :-**

$$I = V \times \rho, \quad V = \text{velocity}, \quad \rho = \text{density}$$

31). **Specific gravity :-**

$$Sp.g = \frac{\text{Density of Substance}}{\text{Density of water}}$$

32). **Pulp Density :-**

$$= \frac{\text{Mass of Pulp}}{\text{Volume of Pulp}}$$

$$= \frac{x}{\text{Volume of water} + \text{Volume of Sand}}$$

$$= \frac{x}{\frac{0.4x}{1000} + \frac{0.6x}{3000}}$$

$1000 \rightarrow$ (water density), $3000 \rightarrow$ (Sand density)

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33). **Bulking factor :-**

$$= \frac{\text{Volume of rock after blasting}}{\text{volume of rock before blasting}}$$

34). **Percentage of Dilution :-**

$$= \frac{\text{Volume of waste rock}}{\text{Volume of insitu ore}} \times 100$$

35). **Energy :-**

Energy = Power × time = V.I.t

$E = RI^2t$ $\quad V = IR$

36). **Cutting speed of drill (V) (m/min) :-**

$V = \pi \times D \times N$

D = dia of rotating drill bit
N = Rotational speed (rpm)

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37). **Powder factor :-**

$$PF = \frac{\text{Volume of coal produced/tonnage factor}}{\text{Mass of explosive required}}$$

38). **Capacity of Dozer :-**

$$Q = 0.6 \times (h)^2 \times L \times \gamma$$

Q = output
h = height of the blade
L = length of the blade
γ = specific weight of the material

39). **Explosive consumption per blast hole :-**

$$= \frac{\pi}{4} \times D_{in}^2 \times LE \times \text{density}$$

40). **Explosive occupied length (LE) :-**

$$LE = \text{Bench height} + \text{Sub grade drilling} - \text{Stemming height}$$

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01). Drilling Calculation :-

(a). Penetration Rate (Percussive) :-

$$= \frac{E.f.T}{A \times SE} \ (m/min)$$

E = the energy per blow (Nm)
f = the no. of blow per minute
T = the rate of transfer of energy
A = the drill hole area (m^2), and
S.E = the specific energy (Nm/m^3)

(b). Penetration Rate (Rotary) :-

$$= \frac{Vc.nt.V}{A} \ (m/min)$$

V = is the volume of rock broken per cutting (m

n = the no. of tooth impacts per revolution

V = the no. of revolution per minute.

A = the area of cross section of hole (m^3)

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02). **Bailing Velocity:-**

$$BV = \frac{V \times 1273406.57}{D^2 - d^2} \ (m/min)$$

BV = the bailing velocity (m/min)
V = the volume of air passed by hammer in DTH drill machine (m^3/min)
D = the hole diameter (m)
d = the diameter of drill tube (m)

03). **Drilling Cost:-**

Dc = (Depreciation + Labour cost + fuel cost + maintance cost) + (Cost of bit × Average drill rate)

$$\text{Depreciation} = \left(\frac{\text{machine cost} - \text{Salvage cost}}{\text{machine life}}\right)$$

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07). Energy produced during blasting :-

$E_{blasting}$ = mass of explosive consumed × rate of heat transfer

= V

mass of explosive consumed = volume of explosive column × density

= W

08). Blasting percentage pull :-

$$BP\% = \frac{\text{Amount of pull}}{\text{Drilled depth}} \times 100$$

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04). **Work Done :-**

Work Done = Force × velocity

$= F \times d$

$d = \frac{\text{Penetration Rate}}{\text{revolution}}$

05). **Explosives occupied Volume :-**

$COV = \left(\frac{\pi d^2}{4}\right) \times L$

d = dia of hole
L = Explosives occupied length

06). **Power generated :-**

$Pg = \frac{\text{Energy generated}}{\text{time } (t)}$

$t = \frac{\text{Length of hole}}{VOD}$

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* Direct haulage :- use in where gradient

$$F = m_t g (i + \mu_t) + m_r g (i + \mu_r)$$

F = Force apply

m_t = mass of tub
g = gravity (9.81)
i = ($\tan\theta = \frac{\text{Perpendicular}}{\text{Base}}$) gradient
μ = (mer) Coefficient of friction
t = tub
r = rope
m_r = mass of rope

* Endless rope haulage :- use in flat

gradient + Friction + Rope

$$F = i(W - w) + \mu(W + w) + 2\mu_1 W_1$$

i = gradient
W = weight of loaded tub
w = weight of empty tub
μ = Coefficient of friction

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* Gravity haulage

$$G_1 = g + g_1 + F + f + f_1$$

G_1 = Gradient force of loaded tub
g = gradient force of empty tub
g_1 = gradient resistance of rope drawing empty tub
F = friction resistance of loaded tub
f = —— " —— " —— empty tub
f_1 = —— " —— " —— rope

$$\therefore \quad G_1 - g - g_1 = F + f + f_1$$

$$iW - iw - iw_1 = uW + uw + uw_1$$

$$i = \frac{uW + uw + uw_1}{W - w - w_1}$$

W = gross wt. of the full tub
w = wt. of empty tub
w_1 = wt. of rope
$F = W \times$ friction (u)
$f = w \times$ friction (u)
$f_1 = w_1 \times$ friction (u_1)

i = Gradient

* Main & tail :-

Tail rope = 2 × main rope (length)

Total rope = 3 × haulage road

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Torque middle $(T_m) = W \times \frac{D}{2}$

Torque Ending $(T_e) = [W - L \times R] \times \frac{D}{2}$

Static Torque $(S_T) = [W - L(B-R)] \times \frac{D}{2}$

B = balanced rope weight

✳ **Energy in Winding :-**

$= mgh$

= load to be raised × Depth

= production rate × Depth

$\text{Kinetic} = \frac{1}{2} \times (\text{unbalanced mass}) \times V_{avg}^2$

(a) $\text{Power} = F \times V \Rightarrow \frac{F \times V}{\text{time}}$

$= \text{Torque} \times w$

w = Angular speed (rad/s)

(b) $W = F \times d$

$\text{Power} = \frac{WD}{\text{unit time}}$

Work done – Joule

Power – J/s

$\text{Velocity } (V) = \frac{2\pi r \times N}{60} = i$

N = revolution (rpm)/min

$V = W \times r$

$W = \frac{2\pi \times N}{60}$

$\text{Radian} = \frac{\text{Arc length}}{\text{Radius}}$

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* Locomotive :-

Tractive effort (TE) = W × μ

Draw bar pull (DBP) = TE − Force req to move to locomotive (Rolling Resistance)

μ = Coefficient of adhesion
W = locomotive weight

* Power :-

power = Force × velocity

* Torque :-

T = Force × distance

Torque beginning (T_B) = $[W + L \times R] \times \frac{D}{2}$

W = mineral weight (Payload)
L = length of shaft
R = Rope weight
D = diameter at drum.

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Friction or Koepe winder :-

$$\frac{T_1}{T_2} = e^{\mu\theta} \quad \leq 1.5$$

T_1 = Tension in rope entering the pulley.
T_2 = Tension leaving the pulley.

μ = Coefficient of friction b/w sheave & rope.
θ = Angle of wrap.

Effective tension = $(T_1 - T_2)$

Power = effective tension × speed

Conveyor belt :-

$$T = abv$$

T = Capacity of conveyor belt (tt/sec)
a = Cross-section area (m^2) = $a = \frac{W^2}{10}$ to $\frac{W^2}{12}$
b = bulk density (tt/m^3)
v = Velocity (m/s)
W = width of belt.

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Pump :-

Theoritical head per stage $(H) = \frac{U(U - V\cot\theta)}{g} \times \eta$

θ = angle of blade tip
U = tangential / peripheral speed of impeller tip
V = radial velocity
η = manometric efficiency
N = Rpm
r = radius of impeller

$$U = \frac{2\pi r N}{60} \text{ (m/s)} \quad \text{or} \quad U = \frac{\pi D N}{60}$$

$$V = \frac{Q}{2\pi r W}$$

W = width of impeller
Q = quantity of flow $(m^3/s) \Rightarrow Q = A \times V$

Area of impeller $(A) = 2\pi r \times W$

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Friction loss due to flowing fluid in a pipe :-

$$H_f = \frac{4flv^2}{2gd} \text{ (m)}$$

f = coefficient of friction.
l = length of pipe.
v = velocity of fluid.
d = dia of pipe.

(V) Velocity of suction = $\frac{Q}{\text{area}}$

(V) Velocity of delivery = $\frac{Q}{\text{area}}$

Reciprocating Pump :-

$$\text{Quantity} = \frac{\pi}{4} D^2 \times S \times N \times \eta \times (\text{no. of ram or piston})$$

D = Diameter of pump piston or ram
S = length of stroke
N = No. of stroke / min (rpm)
η = efficiency Volumetric

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Gear :-

Circumference of pitch circle = πD

Circumferential pitch (Circular) $P_c = \frac{\pi D}{T}$

D = diameter

T = Teeth

Diametrical pitch $(P_D) = \frac{T}{D}$

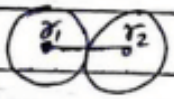
$P_D = \frac{\pi}{P_c}$

$P_{c_1} = P_{c_2}$

$\frac{\pi D_1}{T_1} = \frac{\pi D_2}{T_2}$

$\frac{D_1}{D_2} = \frac{T_1}{T_2}$

Centre to Centre distance



D = module × teeth, $r = \frac{\text{module} \times \text{teeth}}{2}$

$r_1 + r_2 = \frac{m}{2}(T_1 + T_2)$

$\frac{D_1}{D_2} = \frac{T_1}{T_2} = \frac{N_2}{N_1}$ ⇒ Distance travelled by a gear

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Centrifugal Pump :-

$$\text{Head} = \frac{H_1}{H_2} = \frac{N_1}{N_2}$$

H = head
N = rpm

$$\text{Quantity} = \frac{Q_1}{Q_2} = \frac{N_1}{N_2}$$

Q = Quantity
N = rpm

$$\text{pump input} = \frac{PQ}{\eta_{pump}}$$

$$P = \rho g h$$

$$\text{Motor input} = \frac{PQ}{\eta_p \eta_m}$$

P = pressure
h = head
Q = Quantity
η_{pump} = efficiency of pump
η_m = efficiency of motor

$$H = \frac{V^2}{g}$$ Radial bladed

$$\text{Kinetic energy} = \frac{1}{2}mV^2$$ ⇒ pressure energy

$$TE = KE + PE$$

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Power :-

$$\text{Power} = \sqrt{P_s} \times V \times I \cos\phi$$

P_s = Power Supply system

V = Voltage

I = Current

$\cos\phi$ = Power factor

P = power

Drag force :-

$$D_f = C_D \times \frac{1}{2} \rho v^2 \times A$$

C_D = Drag Coefficient

ρ = air density

V = velocity of air

A = area of contact / Cage floor

air density (ρ) = 1.2 Kg/m^3

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* Efficiency of construction :-

$$\eta_c = \frac{Br}{n \times Bw} \times 100$$

Br = Breaking load of rope.
Bw = Breaking load of single wire.
n = no. of wire.

$Br = Sd^2$

$$\text{Nominal FOS} = \frac{\text{Breaking strength of rope}}{\text{Static load (Cage + Rope + Suspension gear)}}$$

$$\text{Actual FOS} = \frac{\text{Breaking strength of rope}}{\text{Static load + Dynamic load}}$$

Dynamic load ⇒ due to acceleration
bending kinetic

$$\text{Space factor} = \frac{N \times \frac{\pi d^2}{4}}{\frac{\pi D^2}{4}} \Rightarrow \frac{Nd^2}{D^2}$$

D = dia. of rope
d = dia of strand
N = No. of strand

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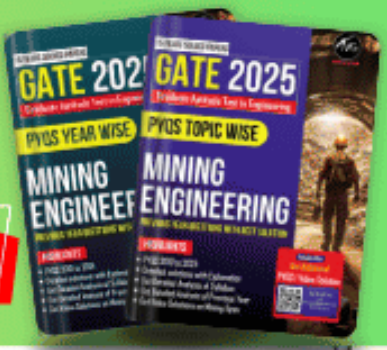

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* Distance covered during acceleration and deceleration :- (use second law of motion of newton).

$$S = ut + \frac{1}{2}at^2$$

S = the displacement of body
u = the initial velocity of a body
t = the time taken by the body to cover displacement
a = constant acceleration of body.

* Simple Gear train :-

$$\frac{N_1}{N_2} = \frac{T_2}{T_1} \quad \left| \quad \frac{N_2}{N_3} = \frac{T_3}{T_2} \quad \right| \quad \frac{N_4}{N_5} = \frac{T_5}{T_4}$$

$$\text{Gear ratio} = \frac{\text{No. of Teeth of driven}}{\text{No of Teeth of driven}}$$

$$= \frac{\text{speed of driven}}{\text{speed of driven}}$$

$$\text{Vehicle Speed} = \frac{\pi DN}{60 \times \text{Gear ratio}}$$

$$\frac{N_1}{N_2} \times \frac{N_2}{N_3} \times \frac{N_3}{N_4} \cdots = \frac{T_2}{T_1} \times \frac{T_3}{T_2} \times \frac{T_4}{T_5} \times \cdots$$

$$\frac{N_1}{N_4} = \frac{T_4}{T_1}$$

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Topic : Factor of safety in Winding and haulage

A. Components of suspension gears	: not less than 10
B. Threaded joint in suspension gears	: not less than 15
C. Winding Rope capping	: not less than 10
D. Winding rope	: not less than 10
D. Haulage rope	: not less than 8
E. Component of coupling in haulage	: not less than 7

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Effective overall % of utilization of shovels

$$= \frac{\text{Actual working hours}}{\text{Total hours}} \times 100$$

$$\text{Availability factor} = \frac{\text{Schedule shift hours} - \text{Maintance hours} - \text{Breakdown hours}}{\text{schedule shift hours}}$$

$$\text{Utilization factor} = \frac{\text{Schedule shift hrs} - \text{Maintanace hr} - \text{Breakdown hr} - \text{Idle hr}}{\text{Schedule shift hours}}$$

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Inverse distance weighted method (IDWM):-

$$\text{Grade} \propto \frac{1}{\text{distance}}$$

$$\bar{x} = \frac{\frac{x_1}{d_1} + \frac{x_2}{d_2} + \cdots + \frac{x_n}{d_n}}{\frac{1}{d_1} + \frac{1}{d_2} + \cdots + \frac{1}{d_n}}$$

Inverse Square method (ISM):-

$$\text{Grade} \propto \frac{1}{D^2}$$

$$\bar{x} = \frac{\frac{g_1}{d_1^2} + \frac{g_2}{d_2^2} + \cdots + \frac{g_n}{d_n^2}}{\frac{1}{d_1^2} + \frac{1}{d_2^2} + \cdots + \frac{1}{d_n^2}}$$

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Sampling Calculation :-

mean grade

$$\bar{g} = \frac{W_1 g_1 + W_2 g_2 + ---- + W_n g_n}{W_1 + W_2 + ---- + W_n}$$

$$\text{mean assay value, } (\bar{g}) = \frac{A_1 g_1 + A_2 g_2 + --- + A_n g_n}{A_1 + A_2 + --- + A_n}$$

$$\text{length, } (\bar{g}) = \frac{l_1 g_1 + l_2 g_2 + ---- + l_n g_n}{l_1 + l_2 + --- l_n}$$

$$\text{Volume, } (\bar{g}) = \frac{g_1 v_1 + g_2 v_2 + --- g_n v_n}{v_1 + v_2 + ---- v_n}$$

$$\bar{g} = \frac{\Sigma g A l}{\Sigma A l} \quad , \quad \bar{W} = \frac{\Sigma A \times W}{\Sigma A}$$

Reserve Estimation :-

Composite grade,

$$\bar{X} = \frac{L_1 X_1 + L_2 X_2 + ---- + L_n X_n}{L_1 + L_2 + ---- + L_n}$$

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* Mine life (By taylor) :-

$$\text{Life} = 0.2\sqrt{\text{Tonnage}}$$

$$\text{Production (mt/day)} = \frac{\text{Tonnage (mineral Reserve)}}{\text{mine life (year)} \times \text{operating days}}$$

Geological reserve > minable reserve > Commercial Reserve

* Geostatistics :-

pair are formed

(a). For h lag distance (n-1) = 3 pairs = $(g_1 - g_2)(g_2 - g_3)(g_3 - g_4)$

(b). For 2h lag distance (n-2) = 2 pairs = $(g_1 - g_3)(g_2 - g_4)$

(c). For 3h lag distance (n-3) = 1 pair = $(g_1 - g_4)$

n = no. of holes
h = lag distance

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* Variogram :-

Linear model, $\gamma(h)^2 = C_0 + C_1 \left(\frac{h}{a}\right)^2$

Spherical model, $\gamma(h)^2 = C_0 + C_1 \left(\frac{3}{2}\frac{h}{a} - \frac{1}{2}\frac{h^3}{a^3}\right)$

where,

C_0 = Chaotic variance (unstructured).
C_1 = Structured variance.
a = range of variance.
h = lag distance.

* Kriging (Covariance) :-

$$Cov(h) = C_0 + C_1 - \gamma(h)$$

h = range of influence.

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Statistical Variance :-

$$\sigma(h) = \frac{1}{m(h)} \sum_{i=1,\, j=2}^{n} (g_i - g_j)^2$$

$m(h)$ = no. of pairs.

$g_i - g_j$ = Difference b/w the grade in pair.

Geostatistical Variance :-

$\gamma(h)$ = Semi Variogram / geostatistical value / variability at h log distance

$\sigma(h)$ = Statistical variance

$$\gamma(h) = \frac{1}{2m(h)} \Sigma (g_i - g_j)^2$$

$$\gamma(h) = \frac{\sigma(h)}{2}$$

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Correlation between γ(H) & H (Covariogram Value)

- Correlation → interdependency between two variables.

$$\text{Correlation} = \frac{\text{Cov}(h)}{\text{Covt}(0)} = \frac{\text{Covariance at } h \text{ of lag distance}}{\text{Covariance at } h=0}$$

$$r = \frac{C_1 + C_0 - \gamma(h)}{C_1 + C_0}$$

Break even Analysis :-

it is a group that shows the profitability of a company at various cost interval.

Break even point :- Indicates neither profits or nor loss.

at BEP :- Total Cost = Total Selling price.

$$FC + V.C \times Q = SP \times Q$$

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Fixed Cost = Total selling price X Quantity − VC × Q

$$Q = \frac{FC}{TSP - VC}$$

FC = Fixed cost (Rs)
VC = Variable cost (Rs/te)
SP = Selling price
Q = quantity (te)
TSP = Total selling price. (Rs/te)

Profit = Total selling price − Total Cost (investment)

* **Mill grade & cut off grade calculation :-**

Grade → % of mineral content in the ore body

mill recovery → % of mineral that can be extracted economical by specific extraction process

Effective grade = grade ore (%) × mill recovery

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$$\text{Grade \%} = \frac{\text{mineral content in ore}}{\text{total ore (waste + ore)}} \times 100$$

$$\text{Dilution \%} = \frac{\text{waste}}{\text{waste + ore}} \times 100$$

Mining & milling cost :-

$$\text{Mining \& milling cost} = \text{Selling price} \times (1-D)g \times R$$

where,

D = Dilution
R = Recovery
g = grade / cut off grade

$$\text{Recovery \%} = \frac{\text{Extracted ore}}{\text{Extracted ore + unextracted ore}} \times 100$$

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01). Blackdamp Calculation :-

Mixture of excess N_2 and CO_2. Sometimes it referred to as chokedamp or stythe.

$$CO_2 \text{ equivalent w.r.t } O_2 = \left(\frac{O_2^r}{O_2^i}\right) \times CO_2^i$$

$$N_2 \text{ equivalent w.r.t } O_2 = \left(\frac{O_2^r}{O_2^i}\right) \times N_2^i$$

where,

O_2^i & N_2^i → are the % of gases in the intake air Sample

O_2^r & N_2^r → are the % of gases in the Return air Sample

$$\text{Blackdamp} = \text{Excess } N_2 + CO_2$$

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$$\text{Concentrate factor, CF} = \frac{\text{Concentrate grade}}{\text{ore grade} \times \text{Recovery}}$$

ore grade < Concentrate grade

$$\text{Mass Recovery} = \frac{\text{Recovery} \times \text{grade (ore)}}{\text{Concentrate grade}}$$

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02). Graham's Ratio :-

It is the ratio of CO produced to O_2 consumed in the mine atmosphere. The ratio indicates the spontaneous heating and its stage.

$$\text{Graham's ratio (in \%) (GR)} = \frac{\text{CO produced}}{\text{oxygen consumed}} \times 100$$

$$= \frac{CO^f - CO^i}{O_2^i - O_2^f} \times 100 \Rightarrow \frac{CO^f - CO^i}{\frac{20.93}{79.04} N_2^f - O_2^f}$$

$$\text{Graham's ratio} = \frac{CO^f - CO^i}{0.265 N_2^f - O_2^f}$$

GR → 0.1 – 0.5% is normal to Coal mine.

GR → 1% indicates existence of Spontaneous heating.

GR → 2% indicates heating in advance stage approaching active fire.

GR – 3% or more indicates active fire.

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- The limits of explosibility of a mixture of combustible gases can be computed from the Le Chatelier equation as:-

04). **Limits of explosibility of a mixture of combustible gases**

$$= \frac{V_1 + V_2 + V_3 + \dots - \%}{\frac{V_1}{x_1} + \frac{V_2}{x_2} + \frac{V_3}{x_3} + \dots}$$

where,

= V_1, V_2, V_3 → Volume of different combustible gases.

x_1, x_2, x_3 → lower limit of explosibility of respective gases.

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Page 5

Q.3). **Respiratory Quotient of breathing :-**

Ratio of Volume CO_2 exhaled and Ratio of Volume O_2 consumed during breathing or respiration.

$$R.Q = \frac{CO_2 \text{ exhaled}}{O_2 \text{ consumed}} = \frac{CO_2^f - CO^i}{O_2^i - O_2^f}$$

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0.6). ## Methane layering

$$CH_4 \text{ layering no.}, L = \frac{U}{\left(4.32 \frac{V}{W}\right)^{1/3}}$$

where,

U → average velocity of air, (m/s).
V → rate of methane emission, (m^3/s).
W → width of airway.

- Layer will not form if —

L > 5 (in case of horizontal roadway).

L > 8 (in case of ascensional ventilation i.e, air moving up the gradient & inclined roadway).

$$CH_4 \text{ layering index} = \left(\frac{24V^2}{C\sqrt{F}}\right)^{1/3}$$

where,

V → average velocity of air.
C → % of methane.
F → cross sectional area of roadway.

$A = L \times b$

$Q = V \times A$

Q = Quantity of air
V = velocity of air
A = CS Area

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- CH_4 → < 0.75% → Return
  < 1.25% → General body of air
- CO_2 → < 0.05%.
- H_2S → 7PPm
- CO → 50PPm
- NO_2 → 5 PPm
- SO_2 → 5 PPm

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05). Emission of methane :-

Amount of fresh air required (Q) :-

$$Q = \frac{100q}{C_p - C_i} - q$$

where,

$q \rightarrow$ emission rate.
$C_p \rightarrow$ Permissible limit.
$C_i \rightarrow$ Gas present in mine.

Or,

CH_4 intake + rate of emission of CH_4 = CH_4 in return

(O_2, CO, H_2S, CO_2) — (O_2, CO, H_2S, CO)

Note :- In the case of O_2 intake — rate of emission of O_2 only sign will change.

(Q × intake) + rate of emission = (Q × return)

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$$NVP = P_1 - P_2 = D\rho_{dg} - D\rho_{ug}$$

D → depth of downcast shaft (m)
g → acceleration due to gravity (m/s^2)
ρ_d → mean density of air in downcast shaft (kg/m^3)
ρ_u → mean density of air in upcast shaft (kg/m^3)

- Density of air changes with temperature (effect of moisture is minor and neglected in usual calculation). If the average temperature (in Kelvin) in the downcast & upcast shaft are T_d & T_u respectively.

- **Density of air in downcast shaft :-**

$$\rho_d = \frac{B_d \times 10^3}{287.1\, T_d} \ (kg/m^3)$$

- **Density of air in upcast shaft :-**

$$\rho_u = \frac{B_u \times 10^3}{287.1\, T_u} \ (kg/m^3)$$

$$\rho = \frac{(B - 0.378e) \times 10^3}{287.1\, T} \ (kg/m^3)$$

e → vapour pressure in Kpa

B_u & B_d are barometric pressure in KPa in upcast and downcast shaft respectively.

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07). Maximum allowable Concentration for Silica :-

$$MAC \text{ of Silica} = \frac{10}{\% \text{ of free silica}} \ (mg/m^3)$$

Que. For indian Coal mines, the maximum allowable Concentration of respirable dust containing 7.5% free silica in (mg/m^3).

Solve:-

$$MAC \text{ of Silica} = \frac{10}{7.5}$$

$$= 1.333 \ mg/m^3$$

08). Natural ventilation pressure (NVP) Calculation from air density :-

Natural Ventilation pressure in a mine is the difference in pressure of air column in the downcast shaft and upcast shaft.

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- MINES RULES 1955
- MINES ACT 1952
- CEAR 2010
- CMR 2017

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For mine shafts in general barometric pressure
$B_u = B_d$ so we can say B Kpa

$$NVP = D.g.\left(\frac{B}{287.1T_d} - \frac{B}{287.1T_u}\right)\times 10^3 \text{ Pa}$$

$$NVP = \frac{g\,DB}{287.1}\left(\frac{T_u - T_d}{T_u . T_d}\right)\times 10^3 \text{ (Pascal)}$$

- Barometric pressure (B) = 101.33 Kpa

09). **Motive Column :-**

The pressure due to Natural ventilation may be expressed in m of air column or motive column, which is the height of an imaginary column of air 1 m^2 in section (at the temp. & pressure of air in the Downcast shaft) that produces a pressure equal to the difference of pressure btw. the bottom of downcast & upcast shaft. it is the effective force causing flow.

$$\text{Motive Column, } h = \frac{NVP}{\rho_d g} \text{ (m)}$$

NVP → in pascal.
ρ_d → the density of downcast shaft in Kg m^{-3}
g → acceleration due to gravity in ms^{-2}

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Resistance of a System of roadways:-

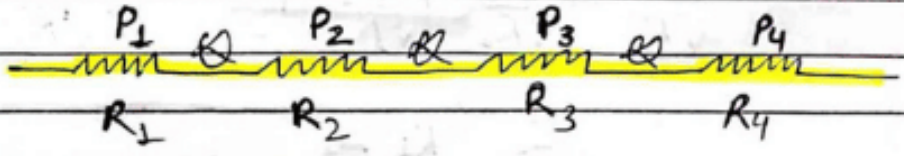
Roadways in Series:-

- when air flows in two or more roadways in series, quantity of air flowing through every roadway is the same, say Q (assuming no leakage) and pressure spent P is the sum of pressures spent in individual roadways.

$$P = P_1 + P_2 + P_3 + \dots + P_n$$

$$RQ^2 = R_1Q^2 + R_2Q^2 + R_3Q^2 + \dots + R_nQ^2$$

$$\boxed{R = R_1 + R_2 + R_3 + \dots + R_n}$$



where,

R → equivalent resistance of n roadway in series.

$R_1, R_2, R_3 \dots R_n$ → are individual resistance of roadways.

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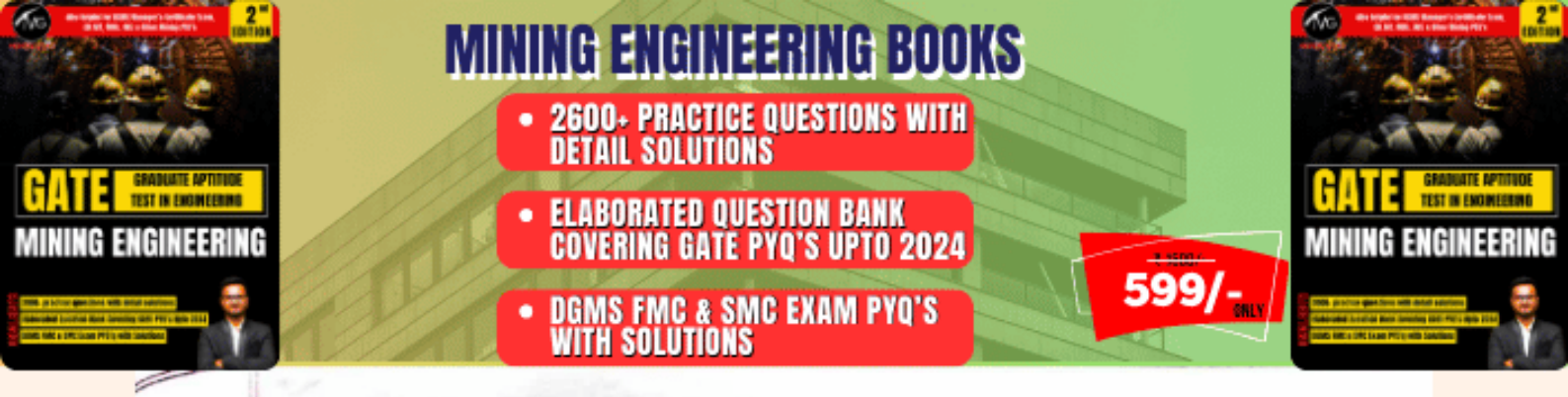

10). **Atkinson's Equation :-**

All the four laws of mine air friction are combined in the following equation, which is known as Atkinson's equation.

Ventilating pressure overcoming friction or pressure loss due to friction.

$$P = \frac{KSV^2}{A}$$

where,

P → Ventilating pressure (in pascal).
K → Coefficient of friction of the airway (Kgm^2 or Ns^2m^{-4})
S → Area of rubbing surface (m^2).
V → Velocity of air current (m/s).
A → Area of cross section (m^2).

$$\text{Quantity of air passing } (Q) = V \times A$$

$$\therefore V = \frac{Q}{A}$$

$$p = \frac{KSQ^2}{A^3}$$

$$S = \text{Perimeter} \times \text{length} \Rightarrow 2(H+W)$$

In case of straight final perimeter $(P) = \pi D$, $A = \frac{\pi}{4} D^2$

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$$h = \frac{T_u - T_d}{T_u} \times D \quad (m)$$

where,

D ⇒ depth of downcast shaft (m).

T_u → temperature of upcast shaft (Kelvin).

T_d → temperature of downcast shaft (Kelvin).

Kelvin = 273 + °C

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Roadways in parallel :-

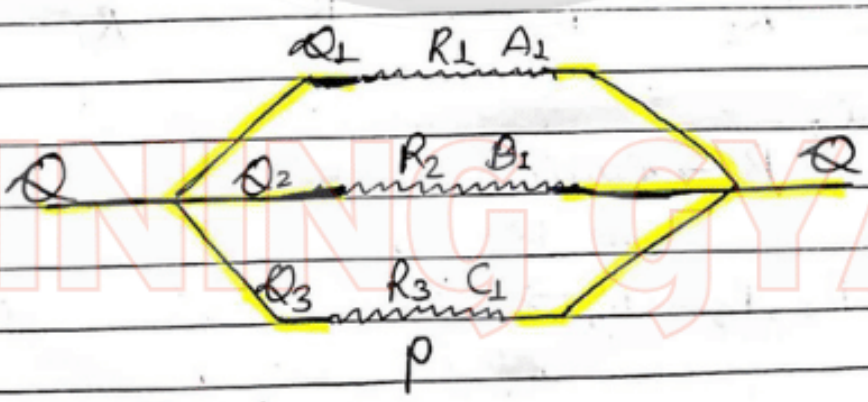
Where flow occurs in two or more roadways in parallel, pressure difference P is the same across every split, while quantity get distributed amongst them depending on their resistance. Thus total quantity of air is sum of individual quantities of air passing through the splits.

$$Q = Q_1 + Q_2 + Q_3 + \dots + Q_n$$

$$\sqrt{\frac{P}{R}} = \sqrt{\frac{P}{R_1}} + \sqrt{\frac{P}{R_2}} + \dots + \sqrt{\frac{P}{R_n}}$$

P – pressure across the split which is same in all

$$\boxed{\frac{1}{\sqrt{R}} = \frac{1}{\sqrt{R_1}} + \frac{1}{\sqrt{R_2}} + \dots + \frac{1}{\sqrt{R_n}}}$$



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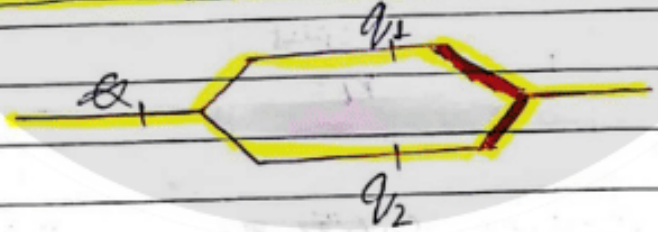
For example:- resistance of 3 roadways in series is 27 times the combined resistance of the same roadways arranged in parallel.

- This shows that splitting offers an economical solution to the problem of increasing the ventilation of the mine.

Let q_1, q_2 are relative quantities in 2 parallel splits NO.1 and NO.2

$$\text{Quantity flowing in split NO.1} = \frac{Q \times q_1}{q_1 + q_2}$$



$$\text{Quantity flowing in split NO.2} = \frac{Q \times q_2}{q_1 + q_2}$$

This method can be used for any number of splits.

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★ if two splits of resistance R_1 and R_2 are in parallel, their combined resistance

$$R = \frac{R_1 R_2}{(\sqrt{R_1} + \sqrt{R_2})^2}$$

★ if three splits of resistance R_1, R_2 & R_3 are in parallel, their combined resistance

$$R = \frac{R_1 R_2 R_3}{(\sqrt{R_1 R_2} + \sqrt{R_2 R_3} + \sqrt{R_3 R_1})^2}$$

★ For n parallel splits of equal resistance

$$R = \frac{R_{split}}{n^2}$$

★ For n series splits of equal resistance

$$R = nR_{split}$$

★

$$\frac{R_{series}}{R_{parallel}} = n^3$$

◆ Resistance of n airways arranged in series is n^3 times the resistance of the same number arranged in parallel.

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* Equivalent orifice of a mine :-

- A fan running at the same speed will not cause the same quantity of air to flow in two different mines because their resistance to flow differ.

$$A = \frac{1.19}{\sqrt{R}}$$

◈ Regulators :-

- Regulator usually rectangular opening left in stopping.
- Regulator helps to reduce flow of air through an airway.
- the size of opening is calculated by equivalent orifice formula.
- these can be permanent, when they are constructed of steels in a concrete stopping (generally made of wooden frame and temporary in nature).
- Air quantity can be adjusted by varying the size of regulator.

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Method 2 :-

if Q_1 and Q_2 are actual quantities flowing in the two splits and Q be the total quantity flowing in the mine.

$$\frac{Q_1}{Q_2} = \sqrt{\frac{A_1^3}{A_2^3} \times \frac{S_2}{S_3}}$$

$$Q = Q_1 + Q_2$$

This method would not be convenient if there are more than two splits in parallel.

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Manometric efficiency of fan :-

The actual depression produced by a fan is somewhat less than the calculated theoretical depression and the ratio of the two is called "Manometric efficiency" of the fan.

$$\text{Manometric } \eta = \frac{\text{Actual depression produced}}{\text{Theoretical depression of the fan}}$$

When power is given to the motor or engine is considered as input, the efficiency is called overall mechanical efficiency.

$$\text{Overall mechanical } \eta = \frac{\text{Air power}}{\text{Power input to fan motor or engine}}$$

But, Air power $= PQ \times 10^{-3}$ KW $\Rightarrow (\rho g H)Q \times 10^{-3}$ Kw

where,

H = head generated by fan, meter.

g = acceleration due to gravity.

ρ = density of air (Standard air density is taken as 1.2 Kg m^{-3}).

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Fan law :-

The three laws given below apply to any fan working on a mine of constant resistance (or equivalent orifice) and hold good for all types of fan :-

if Q → Quality of air
N → Fan speed

Law-1 → Quantity of air delivered varies directly as the fan speed.

$$Q \propto N$$

Law-2 → Pressure produced or W-g developed

$$P \propto N^2 \text{ or } Q^2$$

Law-3 → Power required to drive the fan.

$$HP \propto N^3 \text{ or } Q^3$$

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Pillar Design :-

$$FOS = \frac{\text{Pillar strength}}{\text{Pillar stress}} = \frac{SP}{\sigma p}$$

Pillar stress :- (σp)

- Tributary Area Method :-

$$\sigma p = \frac{\gamma H (wp + wg)^2}{wp^2} = \frac{\gamma H}{1 - R}$$

σp = pillar stress.
γ = unit weight of OB (N/m^3)
H = Depth of pillar (m)
wp = width of pillar, wg = width of gallery

During development on Tributary area extracted,

$$\text{Extraction ratio (R) or recovery ratio} = \frac{(wp+wg)^2 - (wp)^2}{(wp + wg)^2}$$

$$\text{Tributary area (TA)} = (wp + wg)^2 - wp^2$$

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* Square pillar *

* Salamon – Munro Formula :-

$$S_p = 7.2 \times \frac{W^{0.46}}{h^{0.66}} \quad \text{MPa}$$

S_p = strength of pillar
W = pillar width
h = pillar height

* Bieniawski's Formula :-

$$S_p = S_1 \left(0.64 + 0.36 \frac{W}{h}\right) \quad \text{MPa}$$

S_p = strength of pillar, W = pillar width
S_1 = UCS of cubical sample, h = pillar height

* CIMFR Formula :-

$$S_p = 0.27 \sigma_c h^{-0.36} + \frac{H}{160}\left(\frac{W}{h} - 1\right)$$

S_p = strength of pillar (MPa)
σ_c = UCS of cubical sample (MPa)
h = pillar height
H = Depth of seam
W = pillar width

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Pillar width :- (Wp)

$$W_p = W_{c-c} - W_g$$

W_p = width of pillar
W_g = width of gallery
W_{c-c} = centre to centre distance

Pillar strength :- (Sp)

- In-situ coal pillar strength :-

$$S_1 = K l^{-q}$$

K = uniaxial compressive strength of sample
l = length of sample taken
q = constant
S_1 = in-situ coal pillar strength.

$$S_p = K h^{\alpha} w^{\beta}$$

S_p = pillar strength
K = UCS of cubical sample
h, w = pillar height & width
$\alpha = (-1, -0.66)$, $\beta = (0.5, 0.6)$

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Holland & Gaddy formula :-

$$S_p = K \cdot \frac{\sqrt{w}}{h}$$

$K = \sigma_c \sqrt{d}$

d = Cubical Sample side

σ_c = UCS of Cubical Sample of side 'd'

w, h = width and height of pillar

S_p = pillar strength.

Obert - Duvall Formula :-

$$S_p = S_1 \left(0.778 + 0.222 \frac{w}{h}\right)$$

S_p = pillar strength

S_1 = UCS of Cubical Sample

w = width of pillar

h = height of pillar

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* Rectangular pillar *

* Mark-bieniawski's formula :-

$$Sp = S_1\left(0.64 + 0.54\frac{w}{h} - 0.18\frac{w^2}{lh}\right)$$

Sp = pillar strength
S_1 = UCS of sample
l = length of pillar
h = height of pillar
w = width of pillar

* Pressure arch load :-

- Width or span of pressure arch :-

$$W = 0.15H + 60 \text{ (feet)}$$

H = Depth of seam (feet)

- Zone of influence :-

$$ZI = 2\,LTD + \frac{WP}{2} \text{ m}$$

$$\text{Load transfer distance (LTD)} = -1\times10^{-4}H^2 + 0.2701H \text{ m}$$

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* Artificial Support *

* Rock bolt design :-

i). Support of wedge :-

$$FOS = \frac{RF}{DF} \Rightarrow \frac{\text{Resisting force}}{\text{Driving force}}$$

$$FOS = \frac{(W\cos\alpha + T\sin(\alpha+\beta))\tan\phi + C.A}{W\sin\alpha - T\cos(\alpha+\beta)}$$

W = weight $\Rightarrow \frac{1}{2}\gamma h^2 \cot\alpha$ N γ = unit weight

A = area $\Rightarrow \frac{h}{\sin\alpha}$ m^2 h = height

C = Cohesion

ϕ = friction angle

T =

ii). Slot & wedge :-

$$P = \sigma_y . A(\tan\phi\cos\alpha + \sin\alpha)$$

A = area

P = Anchorage (net force applied by bolt)

σ_y = bearing capacity of bolt

$$P = F_s\cos\alpha + F_n\sin\alpha$$

α = Conical angle

F_s = Shear force $\Rightarrow F_n\tan\phi$

F_n = Normal force $\Rightarrow F_s\tan\alpha$

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Force on pillar $(F_p) = \gamma H \times (ZI)^2$

Stress on pillar $(\sigma_p) = \frac{F_p}{A_p}$

Advance pillar Design:→

Active mining Zone (AMZ) $= S\sqrt{H}$

Abutment Zone of influence (D) $= 9.3\sqrt{H}$

H = depth of Seam.

$FOS_{AMZ} = \frac{\text{load bearing capacity}}{\text{Total load}}$

load bearing capacity $= n \times Sp \times A$

n = no. of pillar in AMZ.
Sp = pillar strength
A = pillar area $(l \times b)$

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@. Minimum length of bolt = $\frac{B}{2}\left(\frac{100 - RMR}{100}\right)$ {Bieniawski}

@. Support load = $Y.B\left(\frac{100 - RMR}{100}\right)$ {Bieniawski}

$= Y.B(1.7 - 0.037RMR + 0.0002\,RMR^2)$ {CMRI}

* Secondary Support System *

* Stowing rate = $\frac{\text{Volume of sand} + \text{Volume of water}}{\text{time taken}}$

Concentration of sand (weight) = $\frac{\text{weight of sand}}{\text{total weight of slurry}}$
% in slurry

$= \frac{\text{weight of sand}}{\text{weight of sand} + \text{weight of water}}$

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iii). Expansion shell bolt :-

$$P = \sigma_y \cdot A (\cos 2\alpha + \sin 2\alpha \cdot \tan\phi)$$

iv). Support of bedded rock mass :-

$$FOS = \frac{f\sigma_b (\pi \gamma_b^2)}{(S \times b \times h \times \gamma_{roof})}$$

f = Constant
σ_b = bearing capacity (Pa) of bolt
γ_b = bolt radius (m)
S = Spacing (m)
b = distance between Consecutive bolt (m)
h = height of immediate roof (m)
γ_{roof} = unit weight of roof (N/m^3)

$$\text{Height of immediate roof } (h) = B\left(1 - \frac{RSI}{100}\right)$$

B = Gallery width

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- Constraint :-(1)

$3x_1 + x_2 \leq 300$

Assume : $3x_1 + x_2 = 300$

Put $x_1 = 0$

$3 \times 0 + x_2 = 300$

$x_2 = 300$

$(0, 300)$

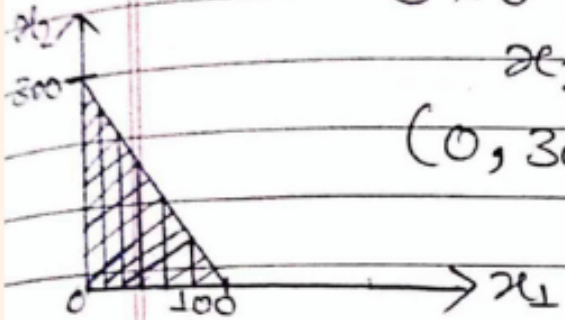
Put $x_2 = 0$

$3x_1 + 0 = 300$

$x_1 = \frac{300}{3}$

$x_1 = 100$

$(0, 100)$



- Constraint :-(2) $x_1 + x_2 \geq 200$

Assume : $x_1 + x_2 = 200$

Put $x_1 = 0$

$0 + x_2 = 200$

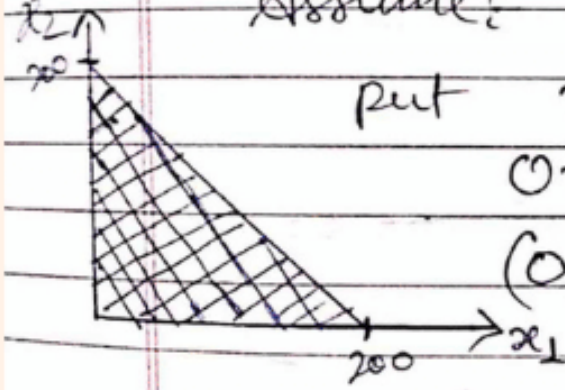
$(0, 200)$

Put $x_2 = 0$

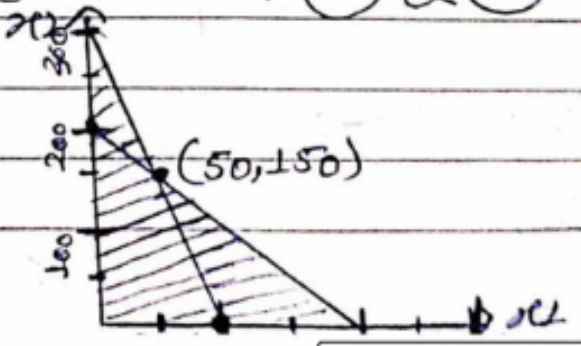
$x_1 + 0 = 200$

$x_1 = 200$

$(200, 0)$



- Combining Constraint (1) & (2)



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System engg.

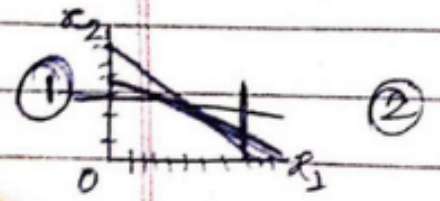
linear programming problem

1). Multiple solution indicate that a problem has multiple optimal solution. (the entire line, edge btw intersection point)

2). unbounded solution indicate that the problem is not well-defined. (the objective goes to infinity)

3). infeasible solution indicate that the problem has no solution (any point outside the feasible region)

4). Unique solution if the problem becomes convex and has a single optimum solution



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Q.4 You make decorative stones for decorating. A ton of stone A requires 2 hours of crushing, 5 hours of sifting, & 8 hours of drying, A ton of stone B requires 6 hours of crushing 3 hours of shifting & 2 hours of drying. The stone A sell for Rs400/ton the stone B sell for Rs500/ton. In a week your plant is capable of 36 hrs of crushing, 30 hrs of shifting, & 40 hrs of drying determine how much of each kind of stones you should make to maximize your revenue.

Maximize = $400x + 500y$

Subject = $2x_1 + 6y_2 \leq 36$ — (1)

$5x_2 + 3y_2 \leq 30$ — (2)

$8x_3 + 2y_3 \leq 40$ — (3)

$x \geq 0$, $y \geq 0$

$2x_1 + 6y_2 = 36$

Put $x_1 = 0$	$y_2 = 0$
$6y = 36$	$2x = 36$
$y = 6$	$x = 18$
$(0, 6)$	$(18, 0)$

$5x_2 + 3y_2 = 30$

Put $x_2 = 0$	$y_2 = 0$
$y = 10$	$x = 6$
$(0, 10)$	$(6, 0)$

$8x_3 + 2y_3 = 40$

Put $x_3 = 0$	$y_3 = 0$
$y = 20$	$x = 5$
$(0, 20)$	$(5, 0)$

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Q.3 Maximize $Z = 4x_1 + 2x_2$

$2x_1 - 2x_2 \leq 20$

$4x_1 \leq 80$

$x_1 \geq 0, x_2 \geq 80$

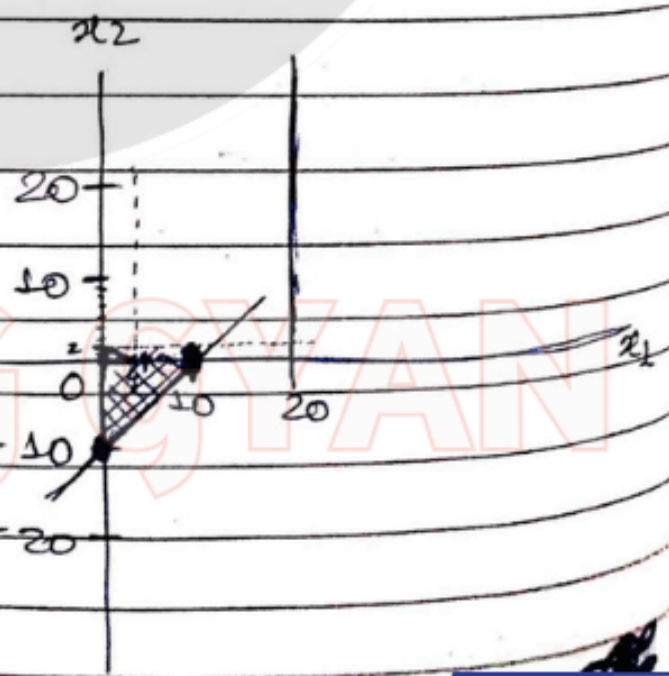
$2x_1 - 2x_2 = 20$

Put $x_1 = 0$	Put $x_2 = 0$
$2(0) - 2x_2 = 20$	$2x_1 - 2(0) = 20$
$-2x_2 = 20$	$2x_1 = 20$
$x_2 = -10$	$x_1 = 10$
$(0, -10)$	$(0, 10)$

$4x_1 \leq 80$

$x_1 \leq 20$

$0, 20$



$4x_1 + 2x_2 \quad -x_1$

Put $x_1 = 0$	$x_2 = 0$
$4(0) + 2x_2 = 0$	$4x_1 + 2(0) = 0$
$2x_2 = 0$	$4x_1 = 0$
$x_2 = 0$	

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Economic order Quantity

EOQ Models

Saathi

Date / /

- C_O = order cost
- C_h = holding cost
- C_I = item cost

* formula :-

* Average inventory = (initial inventory + final inventory) / 2

* Total order cost = (total demand / order quantity) × order cost

* Total holding cost = Average inventory × holding cost

* Total cost of item = Demand × cost of 1 item

* Total Cost = Total order cost + Total holding cost + Total cost of item

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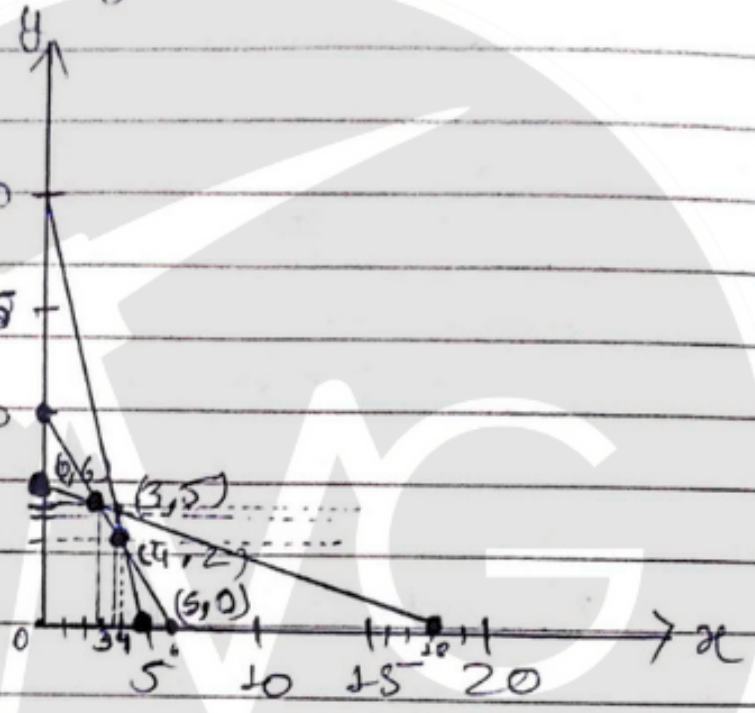
(1) $x = 18, y = 6$

(2) $x = 6, y = 10$

(3) $x = 5, y = 20$



Point = 400x

(0,6) = 3000

(3,5) = 3700 maximum revenue

(4,2) = 2600

(5,0) = 2000

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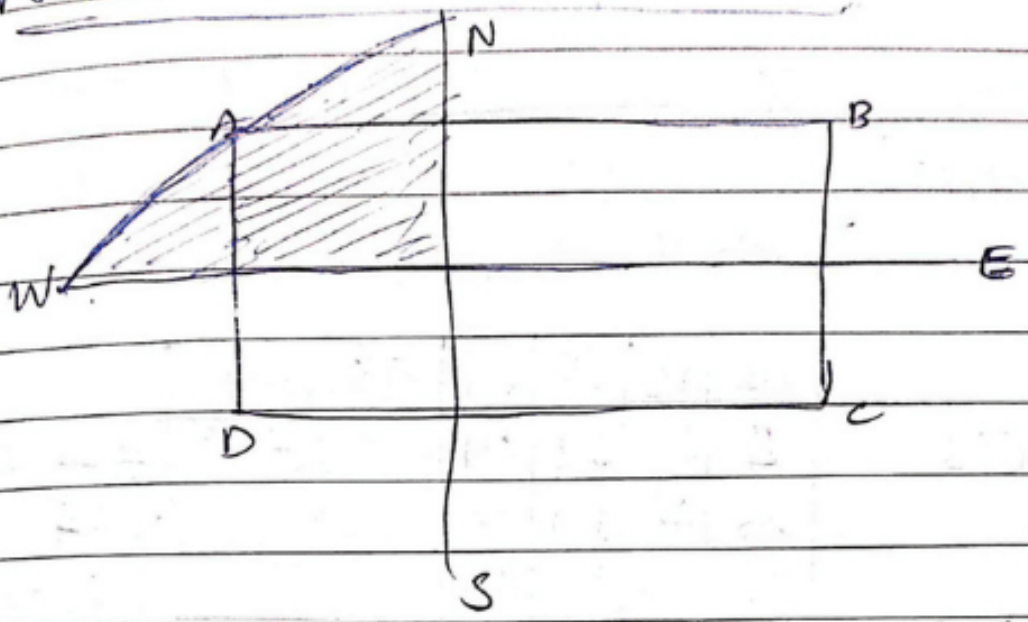
~~Transport model~~

North west corner method :-



Source	Destination				Supply
	N	S	E	W	
A	16	13	22	17	200
B	14	13	19	15	350
C	9	20	23	10	150
Demand	100	140	300	~~150~~ 160	

On this method we allways choose NW value means (Top left value)

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* Economic order Quantity (EOQ) = $\sqrt{\frac{2DC_o}{C_h}}$

D = Demand

EOQ = order Quantity

C_o = order cost

C_h = holding cost

C_I = item cost

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* Assignment problems *

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Date

Minimize problems

	Job1	Job2	Job3	Job4
P1	14	5	8	7
P2	2	12	6	5
P3	7	8	3	9
P4	2	4	6	3

Step-1 :- Substract Row from Smallest No. of each row.

	Job1	Job2	Job3	Job4
P1	9	0	3	2
P2	0	10	4	3
P3	4	5	0	6
P4	0	2	4	1

Step-2 :- Substract Column from Smallest No. of each Column.

	Job1	Job2	Job3	Job4
P1	9	0	3	1
P2	0	10	4	2
P3	4	5	0	5
P4	0	2	4	0

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Transport model

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✱ Least cost method ✱

Date

	D_1	D_2	D_3	D_4	Supply
O_1	11	13	17	14	250
O_2	16	18	14	10	300
O_3	21	24	13	10	400
Demand	200	225	275	250	

On this method we choose always least value.

✱ Vogel approximation method ✱ (VAM)

✱ Penalty = Difference between Smallest and Second Smallest number

	D_1	D_2	D_3	D_4	Supply	
O_1	11	13	17	14	250	⇒ 11 − 13 = 2
O_2	16	18	14	10	300	⇒ 14 − 10 = 4
O_3	21	24	13	10	400	⇒ 13 − 10 = 3
Demand	200	225	275	250		
	5	5	1	4		

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Step-3 :- look for Rows ~~and Columns~~ with Single Zeros.

	Job1	Job2	Job3	Job4
P1	9	0	3	1
P2	0	10	4	2
P3	4	5	0	5
P4	0	2	4	0

$P_1 = Job_2 = 5$

$P_2 = Job_1 = 2$

$P_3 = Job_3 = 3$

$P_4 = Job_4 = 3$

So that, $5+2+3+3=13$

Step-4 :- Make A line Covering all Zeros

Step-5 :- Substract Smallest No. from uncovered numbers

Step-6 :- Add that Smallest No. to Junctions'

JUNCTION = Two line meet.

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Step-(2) :- Substract Row from smallest No. of each Row and after then Colum

	Job1	Job2	Job3	Job4
P1	9	0	3	2
P2	0	10	4	3
P3	4	5	0	6
D	0	0	0	0

P1 = Job2 = 5
P2 = Job1 = 2
P3 = Job3 = 3
D = Job4 = 0

5+2+3+0 = 10

* Assignment Maximization problem :-

	Job1	Job2	Job3	Job4
P1	14	5	8	7
P2	2	12	6	5
P3	7	8	3	9
P4	2	4	6	3

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Unbalanced Demand

Q. 3

	Job1	Job2	Job3	Job4
P_1	14	5	8	7
P_2	2	12	6	5
P_3	7	8	3	9
D	0	0	0	0

we take 'D' as a dummy

Step-1 :- Add Dummy Row / column where supply / job is less.

Page No.

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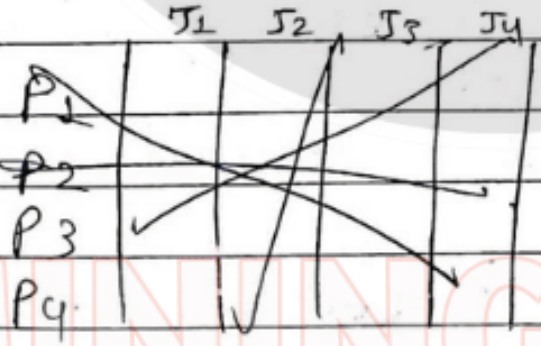
Step-① Subtract each element of matrix from largest NO.

	Job1	Job2	Job3	Job4
P_1	0	9	6	7
P_2	12	2	8	9
P_3	7	6	11	5
P_4	12	10	8	11

Step-② Subtract Row/col from smallest no. of each Row/col.

	J1	J2	J3	J4
P_1	0	9	6	7
P_2	10	0	6	7
P_3	2	1	5	0
P_4	4	2	0	3

Row/col



$P_1 = J_1 = 14$
$P_2 = J_2 = 12$
$P_3 = J_4 = 9$
$P_4 = J_3 = 6$

$14 + 12 + 9 + 6 = 41$

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	T_1	T_2	T_3	T_4
M_1	10	40	60	30
M_2	90	70	100	90
M_3	40	50	110	70
M_4	80	70	80	50

Step-① - Substract Row from Smallest No. of each row.

step-② - Substract column from Smallest No. of each column.

step-③ - Look from Rows with single zeros.

Row

	T_1	T_2	T_3	T_4
M_1	0	30	50	20
M_2	20	0	30	20
M_3	0	10	70	30
M_4	30	20	30	0

Col

T_1	T_2	T_3	T_4	
0	30	20	20	M_1
20	0	0	20	M_2
0	10	40	30	M_3
30	20	0	0	M_4

Step-④ Make a line Covering all zeros.

Step-⑤ Substract Smallest No from unlovered no.

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	T_1	T_2	T_3	T_4
M1	0	20	10	10
M2	20	0	0	20
M3	0	0	30	20
M4	30	20	0	0

Step (5) Add that smallest No. To junction,

	T_1	T_2	T_3	T_4
M1	0	20	10	10
M2	30	0	0	20
M3	0	0	30	20
M4	40	20	0	0

$M_1 = T_1 = 10$
$M_2 = T_3 = 100$
$M_3 = T_2 = 50$
$M_4 = T_4 = 50$

$10 + 100 + 50 + 50 = 210$

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* Queuing System *

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* $P_n = \left(\frac{\lambda}{\mu}\right)^n \left(1 - \frac{\lambda}{\mu}\right)$

P_n = probability of 'n' no of people in the System.

* $L_s = \frac{\lambda}{\mu - \lambda}$

L_s = length of System

λ = mean effective arrival rate (no. of customers arriving per unit time)

μ = mean service rate per busy serve (no. of customers served per unit time)

n = no. of units in the System who are waiting for service and who are being served.

Note:- Arrival rate is always less then the service rate.

* if data show in the case. Arrival rate greater than service. it means is not rate its time.

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Symbol	Definition	Calculation in M/M/1 model
n	Number of customers in a system	
λ	Average arrival rate (e.g., number of arrivals per hour)	
μ	Average service rate per server (e.g., 1/average service time)	
ρ	Operation rate	$\rho = \frac{\lambda}{\mu}$
W_q	Average waiting time in a queue	$W_q = \frac{\rho}{\mu - \lambda}$
W_s	Average entire time in a system	$W_s = \frac{1}{\mu - \lambda}$
L_q	Average number of customers in a queue	$L_q = \frac{\rho\lambda}{\mu - \lambda}$
L_s	Average number of customers in a system	$L_s = \frac{\lambda}{\mu - \lambda}$

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* Critical path method (CPM) *

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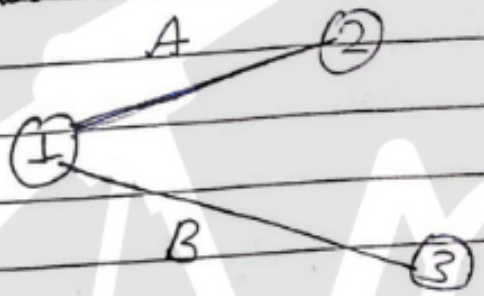
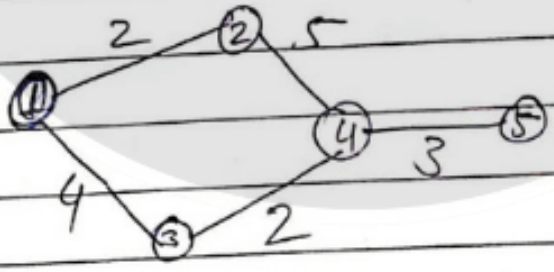
* Critical path - most time taking path
(allocated) (earliest time)



①, ②, ③ = Nodes or event

A, B = activity.

Q.1



Critical path for 1-5

Earliest time 2+5+3 = 10 weeks

the critical path is ①-②-④-⑤

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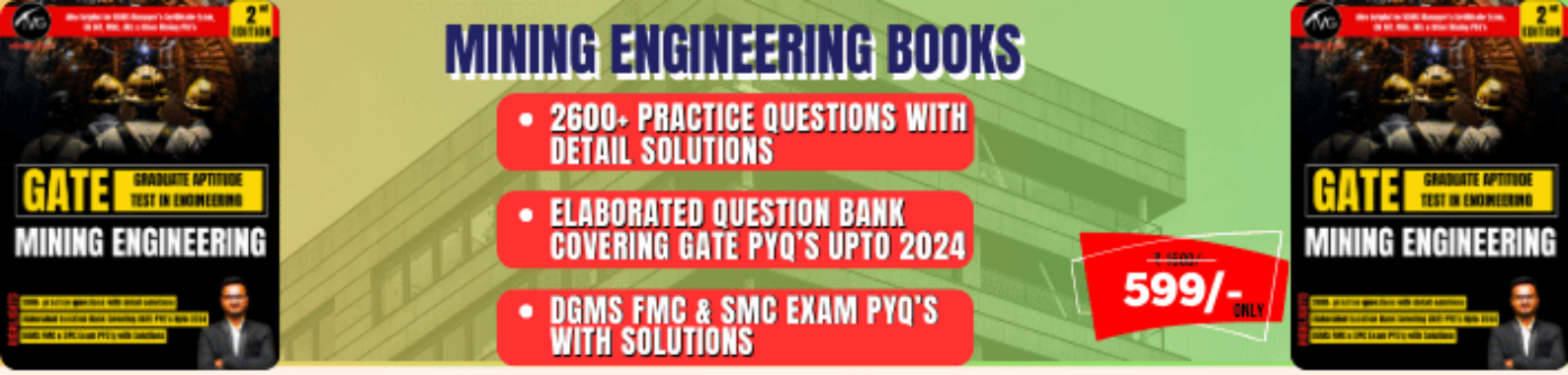

$$P_b = \text{Prob}\begin{bmatrix}\text{both servers}\\ \text{are busy}\end{bmatrix} = \text{Prob}\begin{bmatrix}\text{new arrivals must}\\ \text{wait in queue}\end{bmatrix}$$

$$= \frac{1}{2}\left[\frac{\lambda}{\mu}\right]^2 \left[\frac{2\mu}{2\mu - \lambda}\right] P_o$$

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$$P_o = \text{Prob}\begin{bmatrix}\text{system is}\\ \text{empty (idle)}\end{bmatrix} = 1 - \frac{\lambda}{\mu}$$

$$L_q = \begin{matrix}\text{average number}\\ \text{in the queue}\end{matrix} = \frac{\lambda^2}{\mu(\mu - \lambda)}$$

$$L = \begin{matrix}\text{average number}\\ \text{in the system}\end{matrix} = \frac{\lambda}{\mu - \lambda}$$

$$W_q = \begin{matrix}\text{average time}\\ \text{in the queue}\end{matrix} = \frac{\lambda}{\mu(\mu - \lambda)}$$

$$W = \begin{matrix}\text{average time}\\ \text{in the system}\end{matrix} = \frac{1}{\mu - \lambda}$$

Note:

λ is the arrival rate.

μ is the service rate.

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Project Evaluation and Review Technique (PERT)

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PERT - probabilistic — Time fix nahi hota

PERT - β - distribution
Beta - distribution

Q.1

Optimistic (minimum Time)	Most likely Time (most common time)	Pessimistic (maximum Time)
(a) 1	(m) 2.5	(b) 4

Expected time $(t_e) = \frac{a + 4m + b}{6}$

Standard deviation $\sigma = \frac{b-a}{6}$

$t_e = \frac{1 + 4 \times 2.5 + 4}{6}$

$= \frac{15}{6} = 2.5$

$\sigma = \frac{4-1}{6}$

$= \frac{3}{6}$

$= 0.5$

Probable range to total duration $= T_E \pm 3\sigma$

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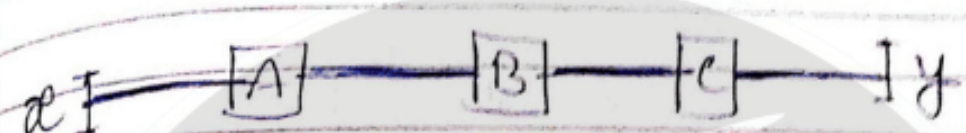
Reliability in Series System

Date



$$R_S = R_A \times R_B \times R_C$$

Q. The failure data analysis of a longwall mining system consisting of three major subsystems revealed that the failure rate of shearer, face conveyor and stage loader are 0.15/hr, 0.2/hr and 0.1/hr respectively. If all the subsystem must be running for a satisfactory system operation, the reliability of the longwall system for a period of 4 hours is

(a). 0.165 (b). 0.259
(c). 0.405 (d). 0.630

$= e^{-(0.15+0.2+0.1)\times 4}$

$= e^{-(0.45)\times 4}$

$= e^{-1.8}$

$= 0.165$

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* Reliability *

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* The general equation of determining the reliability is given by

$$R(t) = e^{-\int_0^t \lambda(t)\,dt}$$

where,
$R(t)$ = is the reliability at time t,
$\lambda(t)$ = is the hazard rate of time-dependent failure rate.
λ = failure rate

$$\text{failure rate } (\lambda) = \frac{\text{no. of failure}}{\text{time}}$$

* Mean time Between failure (MTBF) :-

$$MTBF = \frac{1}{\lambda}, \quad \text{Variance} = \frac{1}{\lambda^2}$$

The reliability or the probability that the equipment run up to time t for exponential distribution failure rate $= e^{-\lambda t} = e^{-t/m}$

where, $m = 1/\lambda$ = mean time between failure (MTBF)

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- Expected time = t_e
- Standard deviation = σ
- Variance = σ^2

Backward Pass → Slack = (Late start) − (earliest start)

or

forward Pass → (Late finish) − (earliest finish)

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* Availability *

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$$\text{Availability} = \frac{MTBF}{MTBF + MTBR} = \frac{F}{F+R}$$

Q.1 MTBF = 4 hr
MTBR = 6 hr

$$\text{Availibility} = \frac{4}{4+6} = \frac{4}{10}$$

$$= 0.4$$

Q.2 Failure Rate = 6/hr
Repair Rate = 4/hr

$$MTBF = \frac{1}{\lambda} = \frac{1}{6} = 0.17, \quad MTBR = \frac{1}{R} = \frac{1}{4} = 0.25$$

$$\text{Availability} = \frac{0.17}{0.17+0.25}$$

$$= 0.4$$

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* Reliability in parallel *

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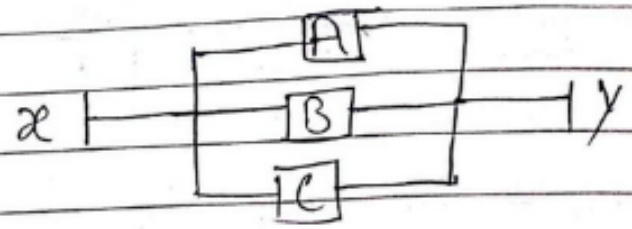

$$R_p = 1-(1-R_A)(1-R_B)(1-R_C)$$

Q.1 A system consists of four elements A, B, C and D which are connected functionally in a parallel configuration. The individual reliability of the elements is 0.80, 0.82, 0.85 and 0.90 respectively. The reliability of the system is

(a). 0.498 (b). 0.602
(c). 0.750 (d). 0.999

$= 1-(1-R_A)(1-R_B)(1-R_C)(1-R_D)$
$= 1-(1-0.80)(1-0.82)(1-0.85)(1-0.90)$
$= 1-(0.2)(0.18)(0.15)(0.1)$
$= 1-(0.00054)$
$= 0.99946 \cong 0.999$

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NOTES

(Vw) Volume of water ⇒ 0
(Ww) weight of water ⇒ 0

then, Total Weight (W) = Ws + ~~Ww~~ = Ws

03). For Saturated Rock :-

Volume of air (Va) = 0

Volume of Void (Vv) = ~~Va~~ + Vw = Vw

Total Volume (V) = Vs + Vv = Vs + Vw

then, Total weight (W) = Ws + Ww

04). Volume :-

$$V = \frac{\pi d^2}{4} \times l$$

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01). 3-Phase of Rock :-

$$V = V_s + V_v = V_s + V_w + V_a$$

V = Total Volume

V_s = Volume of solid
V_w = Volume of water
V_a = Volume of air
V_v = Volume of void

Volume of void (V_v) = $V_a + V_w$

Total weight (W) = $W_w + W_s$

W_w = weight of water
W_s = weight of solid
W_a = weight of air

02). For Dry Rock :-

Volume of voids (V_v) = $V_a + V_w \neq V_a$

Total Volume (V) = $V_s + V_v$
= $V_s + V_a$

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NOTES

$$P = \frac{e}{1+e}$$

$$e = \frac{P}{1-P}$$

* Porosity *

* Void Ratio *

Degree of Saturation (S) :-

$$S = \frac{V_w}{V_v} = \frac{V_w}{V_w + V_a}$$

For dry Rock, $S = 0$

For fully Saturated rock, $S = 1$

Water / moisture Content (w) :-

$$w = \frac{W_w}{W_s} = \frac{M_w}{M_s}$$

Mass :- M

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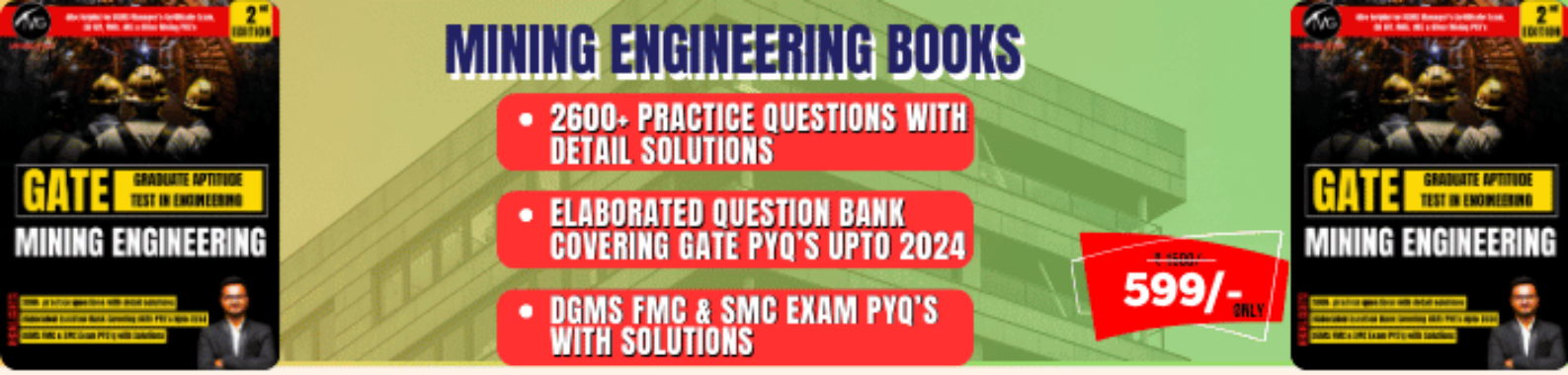

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Weight - Volume Relationships :-

- **Porosity (P) :-**

$$P = \frac{V_v}{V}$$

- **Void ratio (e) :-**

$$e = \frac{V_v}{V_s}$$

- **Relationship between Porosity & Void ratio :-**

$$P = \frac{V_v}{V} \Rightarrow \frac{V_v/V_s}{(V_s+V_v)/V_s} \Rightarrow \frac{(V_v/V_s)}{(1+V_v/V_s)} \Rightarrow \frac{e}{1+e}$$

$$e = \frac{V_v}{V_s} \Rightarrow \frac{V_v/V}{(V-V_v)/V} \Rightarrow \frac{(V_v/V)}{(1-\frac{V_v}{V})} \Rightarrow \frac{P}{1-P}$$

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Density are 3 types :-

- **Bulk density (ρ) :-**

$$\rho = \frac{M}{V}$$

M = mass of porous material
V = per unit volume.

- **Grain density (ρ_s) :-**

$$\rho_s = \frac{M_s}{V_s}$$

M_s = mass of solid
V_s = volume of solid.

- **Dry Density (ρ_d) :-**

$$\rho_d = \frac{M_s}{V}$$

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Density :- $\rho = \frac{Mass}{Volume}$ (kg/m^3)

Specific gravity :- $G_s = \frac{\rho_s}{\rho_w} = \frac{\gamma_s}{\gamma_w}$

Unit weight :- $\gamma = \frac{weight}{volume}$ (N/m^3)

$$\gamma = \frac{W}{V} \Rightarrow \left(\frac{m}{V}\right) g = \rho g$$

g = acceleration due to gravity
$9.81\ m/s^2$

* Density of water (ρ_w) :- $1000\ kg/m^3$
* Unit weight of water (γ_w) :- $9810\ N/m^3$

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* Other Relationships *

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01). $$e = \frac{wG_s}{S}$$

e = Void Ratio
w = moisture Content
G_s = Specific gravity
S = degree of saturation

$$S = \frac{V_w}{V_v} \Rightarrow \frac{V_w}{V_w + V_a}$$

02). $$\gamma = \gamma_d (1 + w)$$

γ = Bulk unit weight
γ_d = dry unit weight
w = moisture content
ρ_d = dry density
ρ = bulk density

Also,

$$\rho = \rho_d (1 + w)$$

03). $$\gamma = \left(\frac{G_s + Se}{1 + e}\right)\gamma_w$$

γ = bulk unit weight
γ_w = unit weight of water
G_s = specific gravity
S = degree of Saturation
e = Void ratio

For dry Rock, $S = 0$

$$\gamma_d = \frac{G_s \cdot \gamma_w}{1 + e} \text{ or } \rho_d = \frac{G_s \cdot \rho_w}{1 + e}$$

For fully Saturated Rock, $S = 1$

$$\gamma_{sat} = \left(\frac{G_s + e}{1 + e}\right)\gamma_w \text{ or}$$

$$\rho_{sat} = \left(\frac{G_s + e}{1 + e}\right)\rho_w$$

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Unit weight are 3 types :-

- Bulk unit weight (γ) :-

$$\gamma = \frac{W}{V} \Rightarrow \rho g$$

- Unit weight of Solid (γ_s) :-

$$\gamma_s = \frac{W_s}{V_s} \Rightarrow \rho_s g$$

- Dry unit weight (γ_d) :-

$$\gamma_d = \frac{W_s}{V} \Rightarrow \rho_d g$$

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Absolute Porosity :-

$$\phi_{abs} = \frac{V_{inter-connected} + V_{isolated\ Pores}}{V}$$

Effective Porosity :-

$$\phi_{eff} = \frac{V_{inter-connected\ Pores}}{V}$$

Darcy's law :-

$$Q = KiA$$

V = Flow velocity (m/s)
K = Hydraulic conductivity (m/s)
Q = Flow rate (m^3/s)
A = Cross-sectional area (m^2)
i = hydraulic gradient (m/m) $= \frac{\Delta h}{L} \Rightarrow \frac{H_A - H_B}{L}$

$$Q = A \times V$$

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Determination of specific gravity and porosity of porous material :-

$$G_s = \frac{W_2 - W_1}{(W_4 - W_1) - (W_3 - W_2)}$$

W_1 = Empty Bottle
W_2 = Empty bottle + Sample
W_3 = Empty bottle + Sample + water
W_4 = Empty + water

$$P = 1 - \frac{\gamma_d}{G_s \cdot \gamma_w}$$

$$\gamma_d = \frac{4(W_2 - W_1)}{\pi D^2 L}$$

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$$K = \frac{kY}{\mu}$$

K = Hydraulic conductivity (m/s).
k = Intrinsic Permeability (m^2).
Y = Unit weight of fluid (N/m^3).
μ = dynamic viscosity (Ns/m^2).

Thermal Conductivity (λ):-

$$q = -\lambda \frac{\Delta T}{L}$$

λ = Thermal Conductivity
q = heat flow (W/m^2 or J/m^3)
$\Delta T = T_2 - T_1$ (Temperature)

$T_2 > T_1$

Thermal diffusivity (a):-

$$a = \frac{\lambda}{\rho c}$$

ρ = density of material/rock.

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Shear modulus (G):-

$$G = \frac{\tau}{\gamma} \qquad G = \frac{(F/A)}{(\Delta L/W)} \Rightarrow \frac{(F/A)}{\tan\theta}$$

τ = Shear Force (F) / CSA (A)

$\gamma = \frac{\Delta L}{W}$

Bulk modulus (K):-

$$K = \frac{\text{Pressure}}{\text{Volumetric strain}} = \frac{-P}{\Delta V/V}$$

ΔV = Change in volume
V = original volume.

$$\text{Volumetric strain} = \frac{\Delta V}{V}$$

$$\frac{\Delta V}{V} \Rightarrow \epsilon_x + \epsilon_y + \epsilon_z \Rightarrow \epsilon_l (1 - 2\mu)$$

$\epsilon_l = \frac{\sigma_x + \sigma_y + \sigma_z}{\epsilon}$, μ = Poisson's ratio.

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Modulus of Elasticity of Rock (Hooke's Law):-

$$E = \frac{\sigma}{\varepsilon}$$

E = elastic content, called modulus of elasticity
ε = strain in the material due to stress applied.
σ = the stress applied on the material

$$(\text{Stress})\ \sigma = \frac{F(\text{load})}{A\ (\text{Area})}\ (N/m^2)$$

$$(\text{Strain})\ \varepsilon = \frac{\Delta L\ (\text{total elongation})}{L\ (\text{original length})}\ (m/m \text{ or } mm/mm)$$

ε = Tensile or compressive strain

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Compressive strength :-

$$\sigma_c = \frac{F}{A} \quad N/m^2 \text{ or } Pa$$

F = load at failure (N)

A = CSA = $\frac{\pi d^2}{4}$ (m^2)

σ_c = uniaxial/unconfined compressive strength.

Tensile strength :-

Direct Tensile test :- $\sigma_t = \frac{F}{A}$ N/m^2 or Pa

Indirect Tensile test (Brazilian Test) :-

$$\sigma_t = \frac{2P}{\pi DL}$$

P = load at failure

D = Diameter

L = length.

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Poisson's ratio :-

$$\mu = \frac{\text{lateral strain}}{\text{longitudinal strain}} = \frac{\Delta D/D}{\Delta L/L}$$

ΔD = Change in Diameter.
D = original Diameter.
ΔL = Change in length.
L = original length.

Relationship between Elastic contants :-

$$E = 2G(1+\mu) = 3K(1-2\mu)$$

E = Young's modulus.
G = Shear modulus.
K = Bulk modulus.

Generalised Hooke's law :-

$$\varepsilon_x = \frac{\sigma_x}{E} - \frac{\mu}{E}(\sigma_y + \sigma_z)$$

$$\varepsilon_y = \frac{\sigma_y}{E} - \frac{\mu}{E}(\sigma_x + \sigma_z)$$

$$\varepsilon_z = \frac{\sigma_z}{E} - \frac{\mu}{E}(\sigma_x + \sigma_y)$$

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Constant Head Test :-

$$Q = \frac{\text{Volume}}{\text{time}} \Rightarrow \text{Flow rate}$$

$$Q = kiA$$

i = hydraulic gradient
K = hydraulic Conductivity (constant head test)
$A = \frac{\pi D^2}{4}$
Volume = water collected in jar
D = Dia of Sample
Δh = head difference in manometer
L = length of Sample
t = time

$$K = \left(\frac{\text{Volume}}{\text{time}}\right)\left(\frac{L}{A\Delta h}\right)$$

Uni-axial Compressive test :-

$$\mu = \frac{-\varepsilon_d}{\varepsilon_l} = \frac{\frac{\Delta d}{d}}{\frac{\Delta l}{l}}$$

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Shear Strength :-

$$\tau = \sigma_n \tan\phi + C$$

ϕ = friction angle
C = Cohesion
σ_n = Stress
τ = Shear strength

Slake durability test (SDI) :-

$$SDI_1 = \frac{m_1}{M} \times 100 \quad , \quad SDI_2 = \frac{m_2}{M} \times 100$$

$$SDI_1 > SDI_2$$

SDI_1 is always more then SDI_2

M = Initial mass of sample.
m_1 = mass retained after 1st cycle.
m_2 = mass retained after 2nd cycle.

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Protodyakonov Test :-

(PSI) protodyakonov impact strength index

$$PSI = \frac{20 \times n}{h} \quad (Rock) \quad (n = 20)$$

$$PSI = \frac{20 \times n}{h} \quad (Coal) \quad (n = 15)$$

n = No. of blows
h = height measured in volumeter of -0.5 mm crushed sample in Cm.

$$\sigma_c = PSI \times 100 \ Kg/cm^2$$

Punch shear test :-

$$\tau = \frac{P}{\pi D L}$$

τ = shear strength
P = load at failure
D = sample dia
L = sample length or thickness.

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Uni-axial Compressive Strength :-

$$\sigma_c = 24 \times I_{s(50)} \text{ MPa}$$

Tri-axial Compression test :-

- Stress at failure Vs confining stress.

$$\sigma_1 = \frac{C_0}{T_0}\sigma_3 + C_0$$

$$C_0 = \frac{2c\cos\phi}{1-\sin\phi}, \quad T_0 = \frac{2c\cos\phi}{1+\sin\phi}$$

$C_0 = \sigma_c \rightarrow$ uniaxial compressive strength,
$T_0 = \sigma_t \rightarrow$ uniaxial Tensile strength.
C = Cohesion.
ϕ = friction angle.
σ_1 = major principal stress.
σ_3 = minor principal stress.

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Point load test :-

- uncorrected point load strength index:

$$I_s = \frac{P}{De^2}$$

∴ P = load at failure, N/m² De = effective diameter

P = Pressure × Ram gauge area

* For Diametric Test *

$$De = D$$

* For Axial Test *

$$De = \sqrt{\frac{4A}{\pi}} \quad \because A = L \times D$$

- Corrected PLSI (For core dia 50 mm) :-

PLSI for 50mm core dia Sample

$$I_{s(50)} = \frac{P}{D^{1.5}\sqrt{50}} \text{ MPa}$$

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Spacing :-

$$S = \frac{\text{length of exposure}}{\text{No. of discontinuities}} \quad \text{m/discontinuity}$$

- It is the perpendicular distance between adjacent discontinuities.
- Frequency = number of discontinuities per meter.

$$\lambda = \frac{1}{S} = \frac{\text{No. of discontinuities}}{\text{length of exposure}} \quad \text{discontinuity/m}$$

Block size index (I_b) :-

$$I_b = \frac{S_1 + S_2 + S_3 + \dots + S_n}{n} \quad \text{m/joint}$$

Volumetric joint count (J_v) :-

$$J_v = \frac{1}{S_1} + \frac{1}{S_2} + \dots + \frac{1}{S_n} = \lambda_1 + \lambda_2 + \dots + \lambda_n$$

Joint/m^3

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S_n = Spacing of nth joint

λ_n = Frequency of nth joint

$$S = \frac{\text{length of exposure}}{\text{No. of joints}}$$

Rock Quality Designation :-

$$RQD = \frac{\Sigma \text{ length of core pieces greater than 10cm}}{\text{Total length of borehole sum}} \times 100$$

Core Recovery :-

$$\text{Core Recovery} = \frac{\Sigma \text{ length of all core piece recovered}}{\text{Total length of borehole sum}} \times 100$$

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RMR Classification :-

- RMR = Rock Mass Rating.
- it is also called geomechanics Classification System.
- it was developed by Z. T Bieniawski in 1976 and later modified in 1989.
- The original RMR (1976) consisted of eight parameters :-

(a). RQD.
(b). State of weathering.
(c). UCS.
(d). Spacing of joints.
(e). Strike and dip.
(f). Separation of joints.
(g). Continuity of joints.
(h). Ground water inflow.

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✳ Relationship between RQD and other parameters

★ ①. $RQD = (100e^{-0.1\lambda}) \times (0.1\lambda + 1)$

λ = Frequency of Joints = $\frac{1}{\text{Spacing}}$

if $\lambda \in (6-16)$

linear approximation

★ ②. $RQD = -3.68\lambda + 110.4$

★ ③. $RQD = 115 - 3.3\,J_v$ (J_v = Volumetric joint Count)

×100

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* Q-System :-

- Q- System = Rock Tunnelling Quality index (Q)
- it was developed by barton et. al (1974) of the Norwegian Geotechnical institute for determination of Rock mass characteristics and tunnel support requirements.
- Q varies in a logarithmic scale from 0.001 to 1000.

$$Q = \left(\frac{RQD}{J_n}\right) \times \left(\frac{J_r}{J_a}\right) \times \left(\frac{J_W}{SRF}\right)$$

RQD = Rock Quality designation, J_a = Joint alteration number.
J_n = Joint set number, J_W = Joint water reduction factor.
J_r = Joint roughness number, SRF = Stress reduction factor.

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- The Modified RMR (1989) considers the following five parameters :-

@. UCS.
ⓑ. RQD.
ⓒ. Spacing of discontinuity.
ⓓ. condition of discontinuity.
ⓔ. Groundwater conditions.
ⓕ. Orientation of discontinuity.

$$RMR = \Sigma(\text{classification parameter}) + (\text{discontinuity orientation adjustment})$$

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Comparison between Q-System & RMR:-

- The common parameter in both RMR and Q-system is RQD.
- RMR includes joint spacing and orientation.
- Q-System includes joint roughness and alteration.
- Q-System also takes into account the rock stress in an excavation and loosening caused by shear zones and faults.

★ $RMR = 9 \log_e Q + 44$ (Bieniawski)

★ $RMR = 15 \log_{10} Q + 50$ (Barton)

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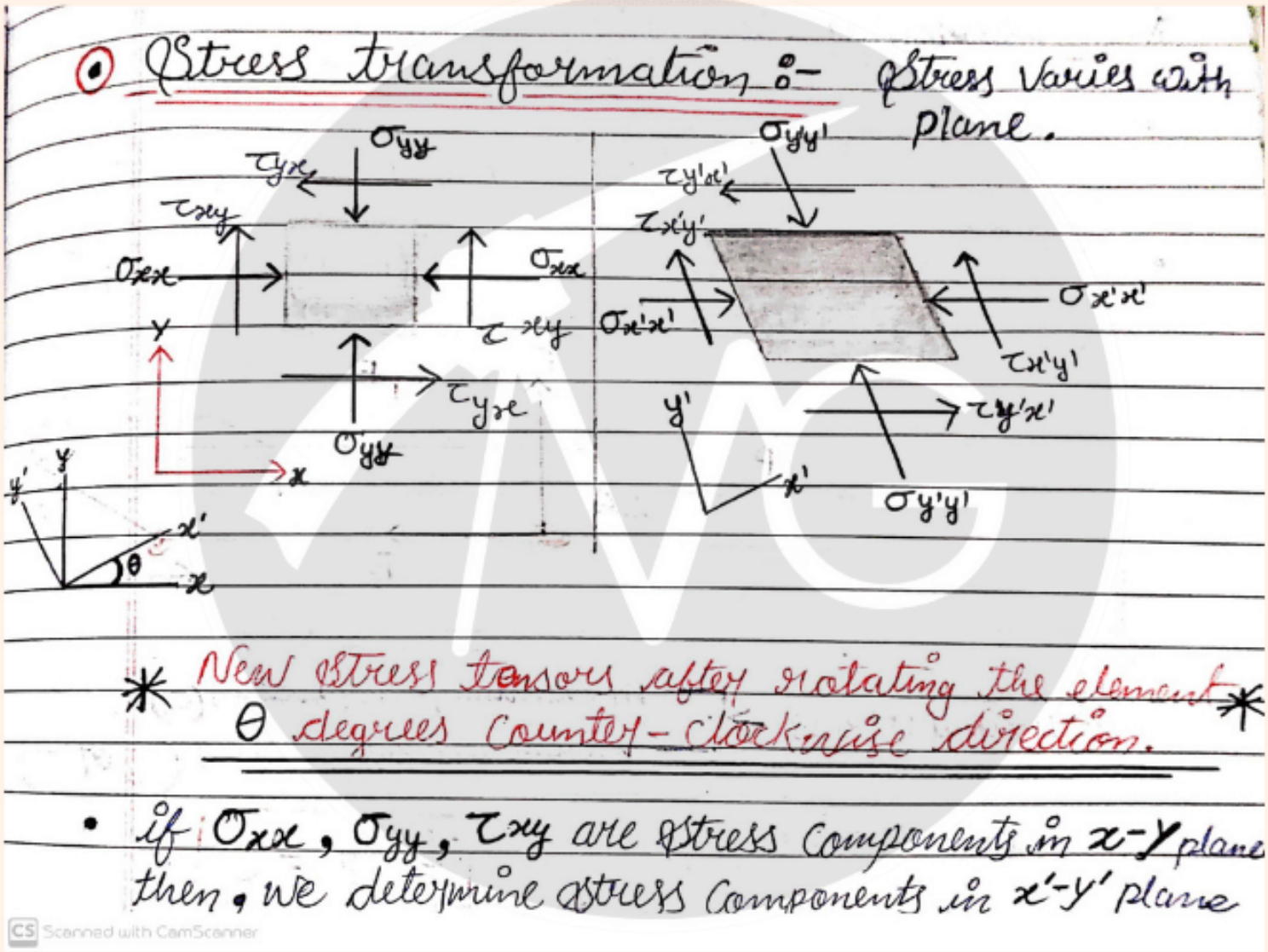

* New stress tensors after rotating the element θ degrees counter-clockwise direction. *

- if σ_{xx}, σ_{yy}, τ_{xy} are stress components in x-y plane then, we determine stress components in x'-y' plane

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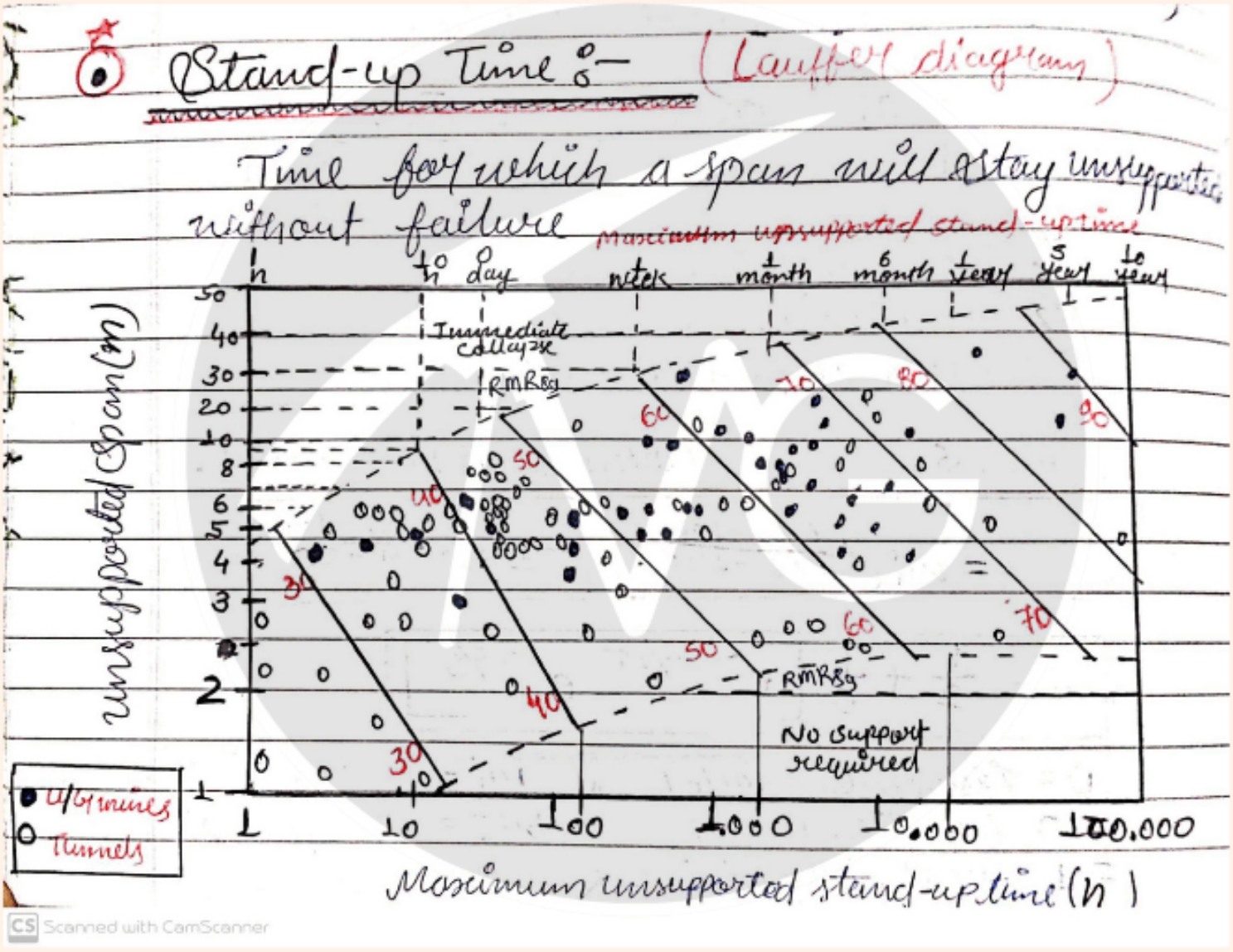

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Comparison of GSI with RMR & Q :-

- RMR and Q-System are dependant upon RQD, so they cannot be applied to very poor rock mass quality.
- GSI is independent of RQD but dependent on geological conditions (degree of jointing, joint condition etc.).

$$GSI = RMR_{89} - 5 \approx RMR_{76}$$

RMR_{89} = 6 parameters (Bieniawski, 1989)

RMR_{76} = 8 parameters (Bieniawski, 1976)

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$\theta_p \rightarrow$ Angle of principal plane

$$\theta_p = \frac{1}{2}\tan^{-1}\left(\frac{2\tau_{xy}}{\sigma_{xx}-\sigma_{yy}}\right)$$

$$\sigma_1 = \left(\frac{\sigma_{xx}+\sigma_{yy}}{2}\right) + \sqrt{\left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)^2 + \tau_{xy}^2}$$

$$\sigma_2 = \left(\frac{\sigma_{xx}+\sigma_{yy}}{2}\right) - \sqrt{\left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)^2 + \tau_{xy}^2}$$

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→ $\sigma_{x'x'}$, $\sigma_{y'y'}$, $\tau_{x'y'}$

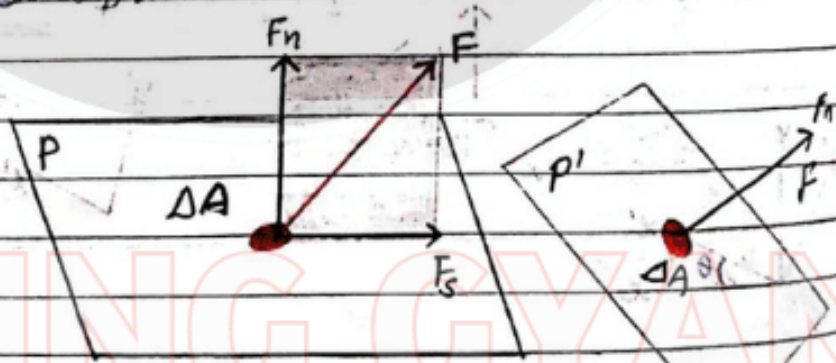
$$\sigma_{x'x'} = \left(\frac{\sigma_{xx}+\sigma_{yy}}{2}\right) + \left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)\cos 2\theta + \tau_{xy}\sin 2\theta$$

$$\sigma_{y'y'} = \left(\frac{\sigma_{xx}+\sigma_{yy}}{2}\right) - \left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)\cos 2\theta - \tau_{xy}\sin 2\theta$$

$$\tau_{x'y'} = -\left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)\sin 2\theta + \tau_{xy}\cos 2\theta$$

Principal Stress :-

- The plane in which shear stress acting is zero, is called principal plane.
- The normal stress on such a plane is called principal stress.



Principal Stress → maximum principal stress (σ_1)
minimum principal stress (σ_2 or σ_3)

on a principal plane, $\tau_{x'y'} = 0$

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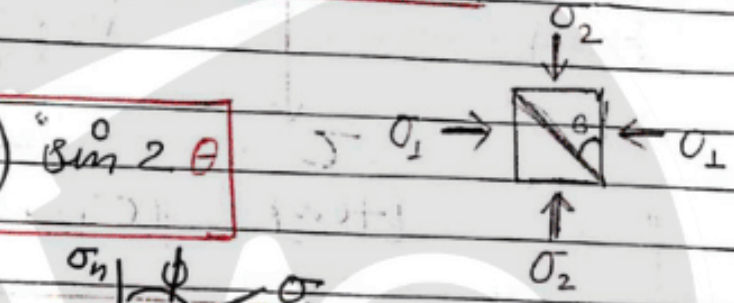
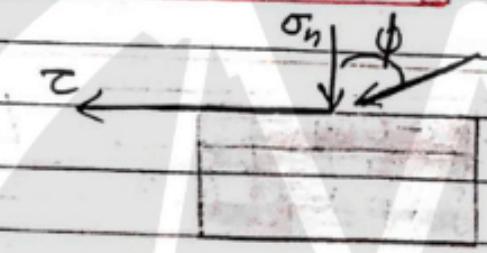
* Principal stress :-

$$\sigma_n = \left(\frac{\sigma_1 + \sigma_2}{2}\right) + \left(\frac{\sigma_1 - \sigma_2}{2}\right)\cos 2\theta$$

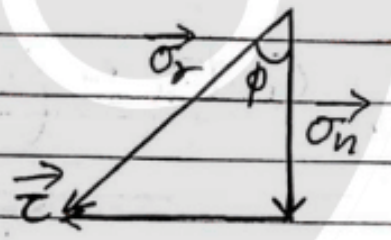
$$\tau = \left(\frac{\sigma_1 - \sigma_2}{2}\right)\sin 2\theta$$





$$\sigma_r = \sqrt{\sigma_n^2 + \tau^2}$$

$$\phi = \tan^{-1}\left(\frac{\tau}{\sigma_n}\right)$$



σ_r = Resultant stress ($\vec{\sigma_r} = \vec{\sigma_n} + \vec{\tau}$)

ϕ = friction angle / Internal angle of friction

* Mohr circle :-

Represent biaxial stress field on $\tau - \sigma_n$ plane.

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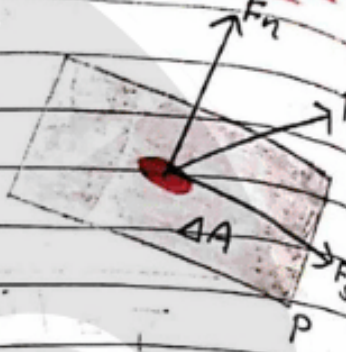
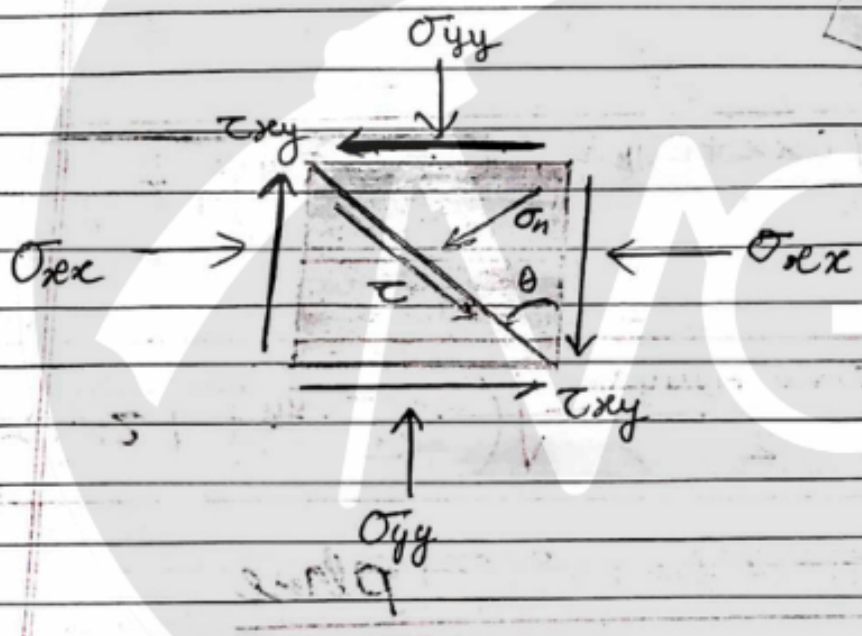
Normal and shear stress for biaxial field :-

$$\sigma_n = \lim_{\Delta A \to 0} \frac{F_n}{A}, \quad \tau = \lim_{\Delta A \to 0} \frac{F_s}{A}$$





$$\sigma_n = \left(\frac{\sigma_{xx}+\sigma_{yy}}{2}\right) + \left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)\cos 2\theta + \tau_{xy}\sin 2\theta$$

$$\tau = \left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)\sin 2\theta - \tau_{xy}\cos 2\theta$$

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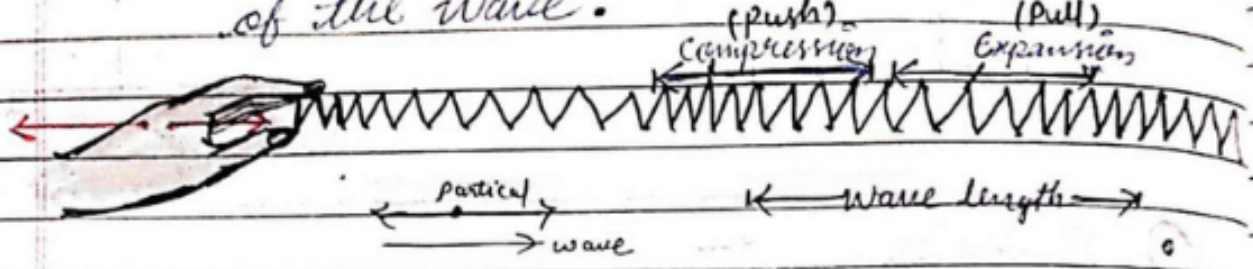
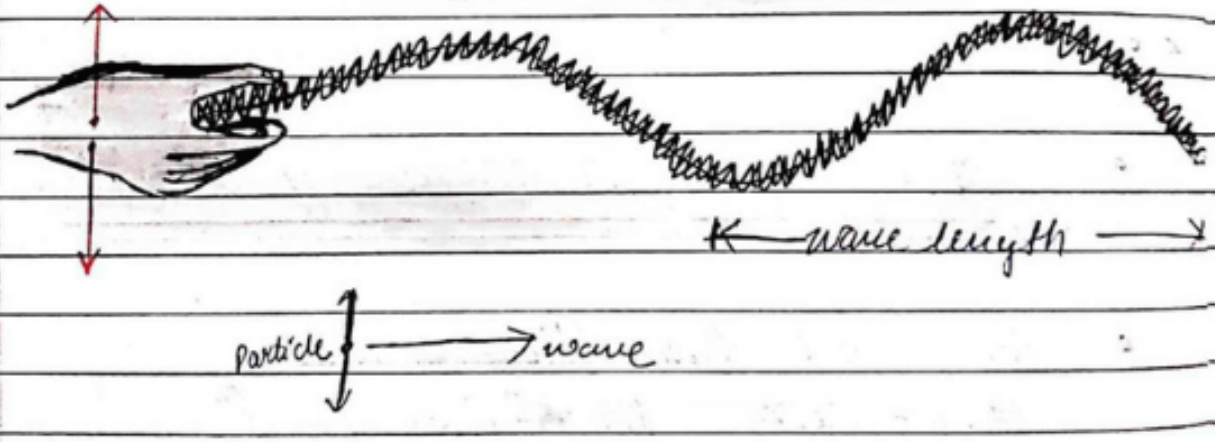
* Longitudinal and Transverse waves :-

- **Longitudinal :-** They cause the medium to move parallel to the direction of the wave.



- **Transverse :-** They cause the medium to move perpendicular to the direction of the wave.



- **Velocity of longitudnal wave :-**

$$V_c = \sqrt{\frac{\epsilon}{\rho}}$$

ϵ = Elastic modulus.
ρ = density of medium.

- **Velocity of Transverse wave :-**

$$V_t = \sqrt{\frac{G}{\rho}}$$

G = Shear modulus.
ρ = density of medium.

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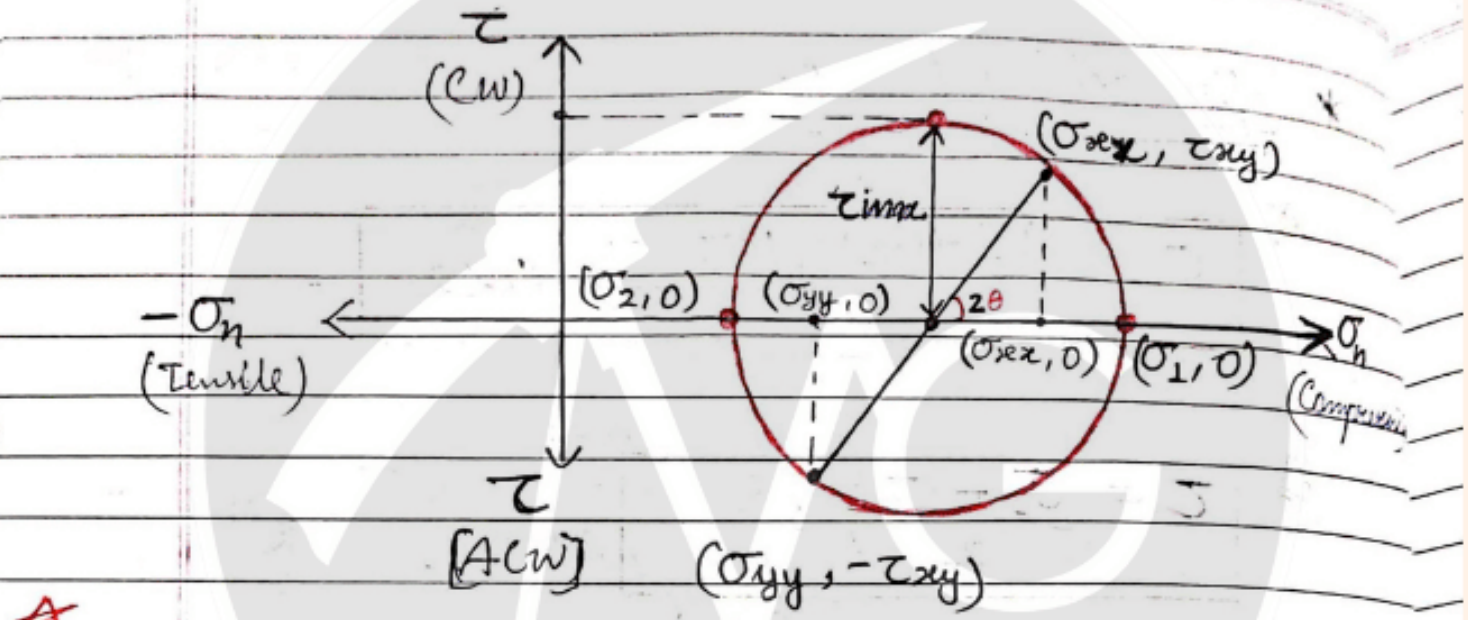

★

① Radius :- $\sqrt{\left(\frac{\sigma_{xx}-\sigma_{yy}}{2}\right)^2+\tau_{xy}^2} = \left(\frac{\sigma_1-\sigma_2}{2}\right) = \tau_{max}$

② Centre :- $\left(\frac{\sigma_{xx}+\sigma_{yy}}{2}, 0\right) = \left(\frac{\sigma_1+\sigma_2}{2}, 0\right)$

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S-Waves:-

- 'S-wave' = Secondary wave or shear wave.
- They are slower than P-wave and are second to arrive at the seismic station.
- They cannot move through liquids.
- As they can move through solids only, this led to the conclusion that earth's outer core is liquid.
- A particle subjected to S-wave move in the direction perpendicular to the propagation of wave, so they are transverse waves.
- They move the rock in an up and down motion.

$$V_S = \sqrt{\frac{G}{\rho}}$$

G = Shear modulus.
ρ = density of medium.

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P-Waves:-

- 'p-wave' = primary wave or pressure wave.
- They are also known as compressional wave because they push and pull.
- They are the fastest wave and are the first to arrive at a Seismic Station.
- They can move through solid rock and fluids, like water or the liquid layers of the Earth. (mantle).
- A particle subjected to p-wave move in the direction of propagation of wave, so they are longitudinal waves.
- They stretch the rock through which they move.

$$V_p = \sqrt{\frac{\frac{4}{3}\mu + K}{\rho}}$$

μ = Shear modulus
K = Bulk modulus
ρ = density of medium

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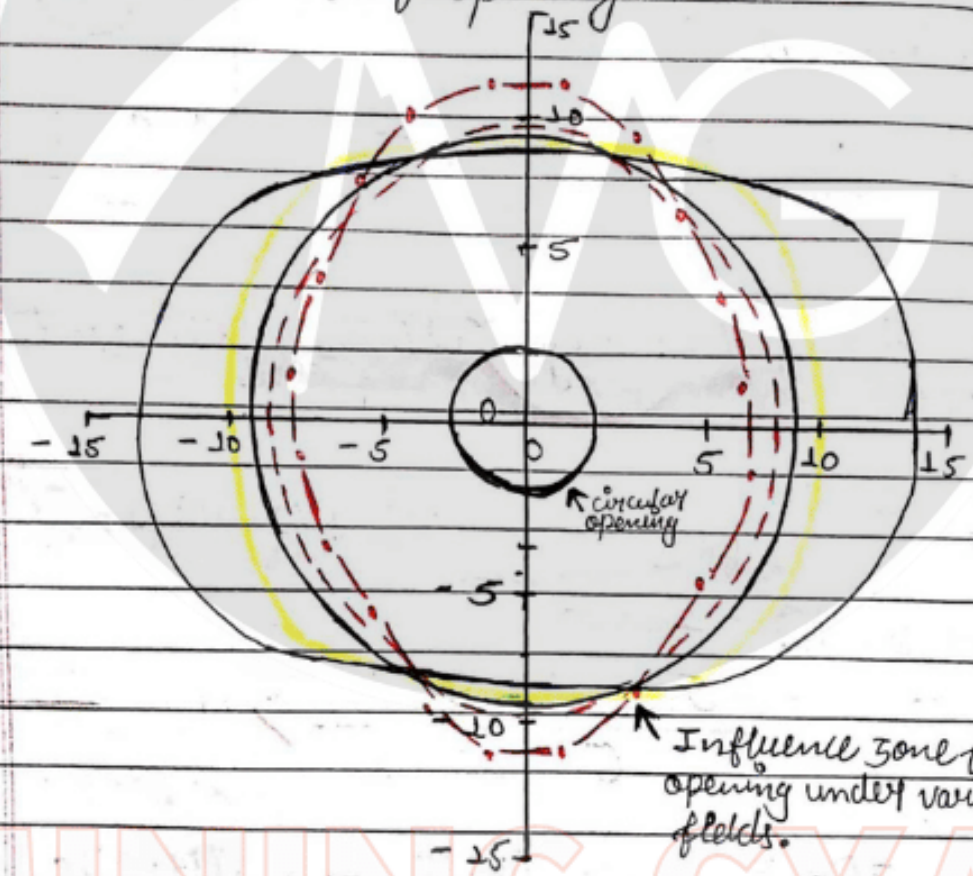
Put ② & ③ in eq. ①.

$$(1+K)\left(\frac{a^2}{r^2}-0.01d\right)+(1-K)\cos 2\theta\left(\frac{3a^4}{r^4}-0.01d\right)=0$$

put [d=5 , K=1]

$$r_{ZOI} = 9\sqrt{20}$$

a = Radius of opening.



K=0.5 — K=0.75 — K=1.0 — K=1.5 — K=

K < 1 Vertical ellipse
K = 1 Circle ($r_{ZOI} = a\sqrt{20}$)
K > 1 Horizontal ellipse

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Kirsch equations (for induced stress) :-

(1). $$\sigma_{rr} = \frac{P_0}{2}\left\{(1+K)\left(1-\frac{a^2}{r^2}\right)-(1-K)\left(1-\frac{4a^2}{r^2}+\frac{3a^4}{r^4}\right)\cos 2\theta\right\}$$

(2). $$\sigma_{\theta\theta} = \frac{P_0}{2}\left\{(1+K)\left(1+\frac{a^2}{r^2}\right)+(1-K)\left(1+\frac{3a^4}{r^4}\right)\cos 2\theta\right\}$$

(3). $$\tau_{r\theta} = \frac{P_0}{2}\left\{(1-K)\left(1+\frac{2a^2}{r^2}-\frac{3a^4}{r^4}\right)\sin 2\theta\right\}$$

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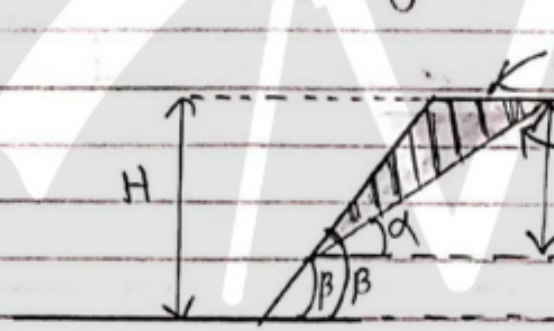
1: Plane failure:-

- In this failure mode, the sliding block rests on a discontinuity plane and it slides, when shear stress acting on the discontinuity plane becomes greater than its shear strength.
- The discontinuity surface is more or less planar or gently undulating.



α = angle of discontinuity plane.

β = Bench angle / Slope angle

h = height of sliding block

H = Bench/slope height

$$FOS = \frac{\text{Shear strength}}{\text{Shear stress}} = \frac{\text{Resisting force}}{\text{Driving force}}$$

MC criteria :- $\tau = \sigma_n \tan\phi + C$

$$\sigma_n = \frac{F_n}{A} = \frac{W\cos\alpha}{A}, \quad \tau = \frac{F_s}{A} = \frac{W\sin\alpha}{A}$$

RF = (Shear Strength) × A, DF = (Shear stress) × A

$$FOS = \frac{(W\cos\alpha)\tan\phi + CA}{W\sin\alpha}$$

W = weight of sliding block.
A = Area of sliding surface.
α = Angle of discontinuity plane
C = Cohesion
ϕ = friction angle

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Estimation of maximum Subsidence:-

- The maximum possible Subsidence (S_{max}) at the middle of the trough is directly related to the height of the excavation (or mining height), properties of OB strata and level of backfilling or stowing.
- For a longwall panel having seam dip of α, the maximum possible Subsidence is

$$S_{max} = a\,m\cos\alpha$$

α = dip of seam.
a = Subsidence factor.
m = mining height.

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Excavated Zone (0m) → Immediate roof (5-10m) → Main roof (20-30m) → Subsidence

Bulking Factor :-

- It is the ratio of broken rock mass to that of in-situ rock mass.

$$K = \frac{\text{Volume of rock after caving}}{\text{Volume of rock before caving}}$$

K = Bulking factor

$$K = \frac{H}{h-1}$$

H = mining height
h = caving height

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$RF = \tau \times A$
$= \tau \times (l \times 1)$
$= \tau l$
$= cl$

$DF = W$

$W = \gamma \cdot v = \gamma A$

Area of sector $= \frac{\pi R^2 \theta}{360}$

Resisting moment $= cl \times R$
$= cR^2\theta$

Driving moment $= Wx$

$$FOS = \frac{cR^2\theta}{Wx}$$

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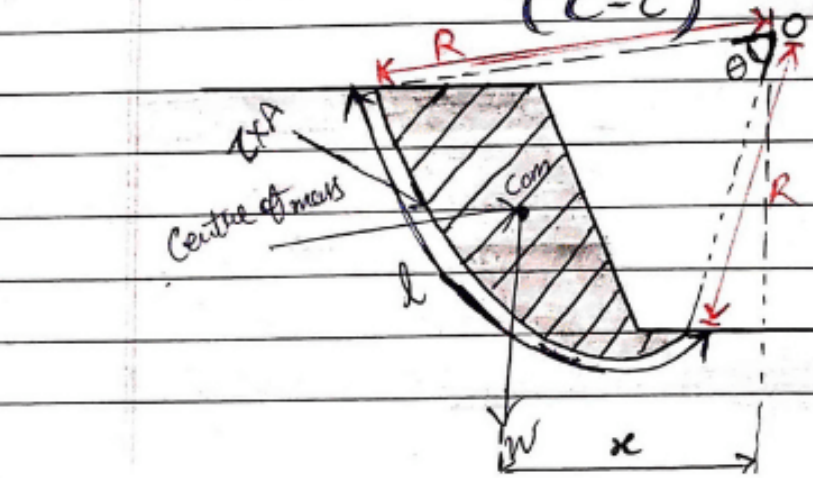
FOS for Circular failure:-

Assumption:- undrained shear strength ($\phi = 0$)

($\tau = c$)



R = Radius of slip circle

θ = Angle subtented by arc at centre.

W = weight of slip circle

x = distance between COM & centre of slip circle.

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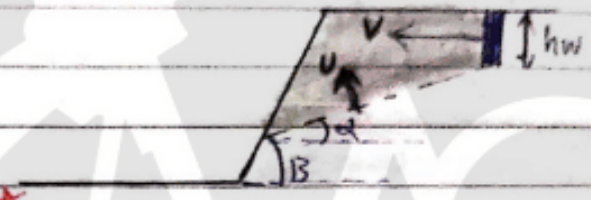
Case-V :- Tension crack with water -



$$V = \frac{1}{2}(\rho_w) h_w^2 \qquad U = \frac{1}{2}(\gamma_w h_w) \times A \quad N$$

$$FOS = \frac{(W\cos\alpha - V\sin\alpha - U)\tan\phi + C.A}{(W\sin\alpha + V\cos\alpha)}$$

$\rho_w = 1000 \text{ Kg/m}^2$

$\gamma_w = 9810 \text{ N/m}^3$

h_w = depth of tension crack with water.

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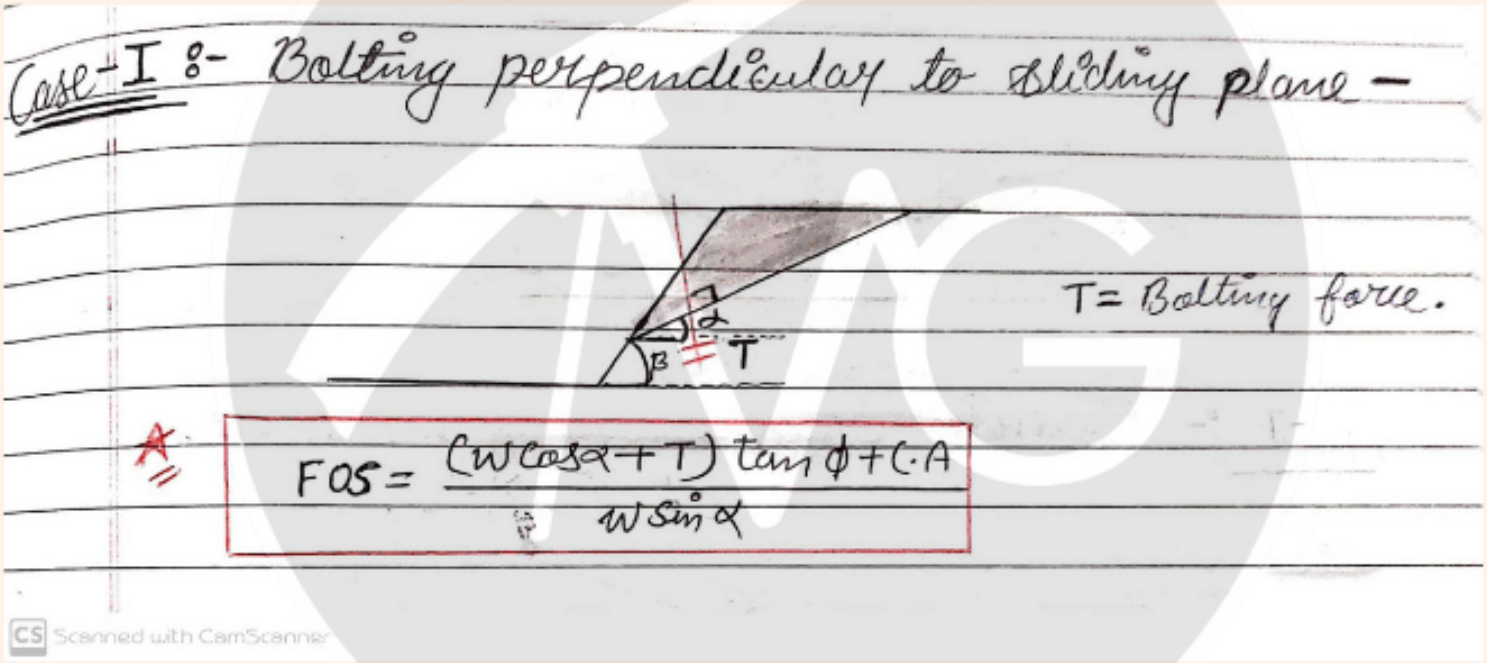

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